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DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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MINING METHODS AND COSTS AT TINTIC STANDARD MINE, TINTIC DISTRICT, UTAH

BY

JAMES W. WADE

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By James W. Wade[2]

INTRODUCTION AND ACKNOWLEDGMENTS

This paper describing the mining practices at the Tintic Standard mine is one of a series being prepared by the United States Bureau of Mines.

In the preparation of this paper each subject has been presented by the engineer in charge. L. R. Dobbs wrote the part pertaining to the technique of square-set mining, J. M. Snow the part on geology, and M. D. Paine, the part on ventilation. All of the drawings are the work of H. W. Waterfield of the engineering department. The statistics of the Tintic district were prepared by A. G. MacKenzie. Each writer has attempted to show the difficulties that arise in operating and the practices used to overcome them.

The writer begs the indulgence of those to whom these elementary things are well known and practiced, in order that less-experienced men may profit. Practically every miner knows what a square set is, yet many do not understand the advantages that may lie with the differences in framing, how to block a set properly, how to place a stull from a set to a loose slab near by, how to block with stringers when mining adjacent to high gobs, or how to take the necessary care in cleaning out a section preparatory to filling.

HISTORY OF DISTRICT

The Tintic mining district is in Juab County, Utah, about 60 miles due south of Salt Lake City and is served by the Denver & Rio Grande Western and the Union Pacific Railroads and by an improved state highway (fig. 1). The first ore discoveries were made in 1869 at outcrops in the western and southern parts of the district. Production was limited by lack of metallurgical facilities until the late seventies. From that time on the Salt Lake Valley smelting companies became important factors in the district's growth; as the smelting facilities improved, giving more favorable rates and recoveries, the production of the district has expanded until a total yearly production of 420,000 tons has been attained. Since its beginning, the Tintic district has produced $324,750,864.00 in gold, silver, lead, copper, and zinc and paid $51,968,619.00 in dividends.

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[1] - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6360."

[2] - One of the consulting engineers, U. S. Bureau of Mines, and Assistant Manager, Tintic Standard Mining Co.

From 1918 to the present the principal producing mines have been in the eastern part of the district. In this part of the district the development was deferred, because no well-defined outcrops indicative of mineral were recognized. No ore bodies outcropped. The top of commercial ore bodies was found from 500 feet below the surface downward. Such lack of outcrop delayed the discovery and development until the early part of the 1900's; then the Colorado and Iron Blossom mines were found. Discovery of the Tintic Standard mine in 1916 followed. Then came in rapid succession, the Eureka Standard, North Lily, and Eureka Lilly mines. Figure 2 is a property map of the district.

The development of the Tintic Standard was begun by E. J. Raddatz in 1907, and his work in the development of that mine was the forerunner of a new epoch in the annals of the Tintic mining district.

GEOLOGY OF TINTIC MINING DISTRICT AND TINTIC STANDARD MINE

From early Cambrian throughout the entire Paleozoic and into the Mesozoic time, there was a long period of continuous deposition of sedimentary rocks. These deposits finally reached a great thickness, probably 25,000 or 30,000 feet, consisting of quartzite, shales, and limestones.

Following this period of deposition and at about the end of Jurassic time the entire section was uplifted and thrown into a series of folds and faults from compressive forces. From this great movement there resulted in Tintic a huge synclinal fold, the axis of which extends north and south, dipping slightly to the north. In the west limb of this syncline the beds stand almost vertically and in places are slightly overturned until they have a steep westerly dip; in the easterly limb the beds are flatter, having a westerly dip of only 20 to 40°.

The forces that caused this syncline also resulted in much faulting. Many of these faults were along the bedding planes with north-south strikes, and others were transverse to the bedding. The vertical throw was considerable, in some instances reaching as much as 1,500 feet. This folding and faulting fractured the beddings and created loose open channels which later were readily accessible to mineral bearing solutions.

During the period of folding and faulting and for a long time afterwards erosion forces were active, and a great deal of the sedimentation was carried away, leaving the surface topography of the Tintic district comparatively rugged, with several high peaks and ridges, such as Eureka Peak and Pinion Peak. After this period of erosion there remained exposed some 6,000 feet of Cambrian quartzite and shales and about 7,000 feet of Paleozoic limestones.

The next and most important period influencing the ore deposits of this district was one of great volcanic activity, which began about early

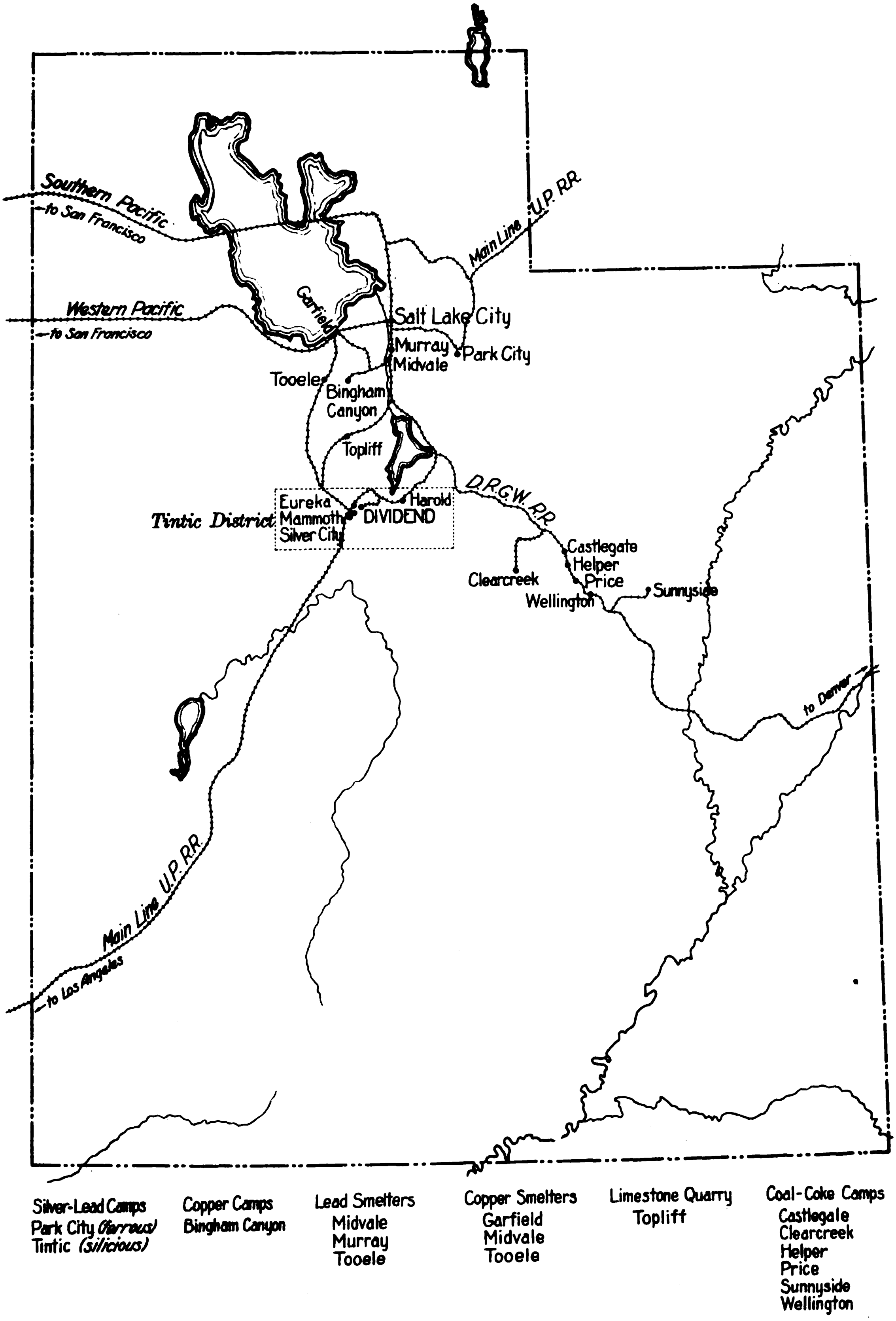
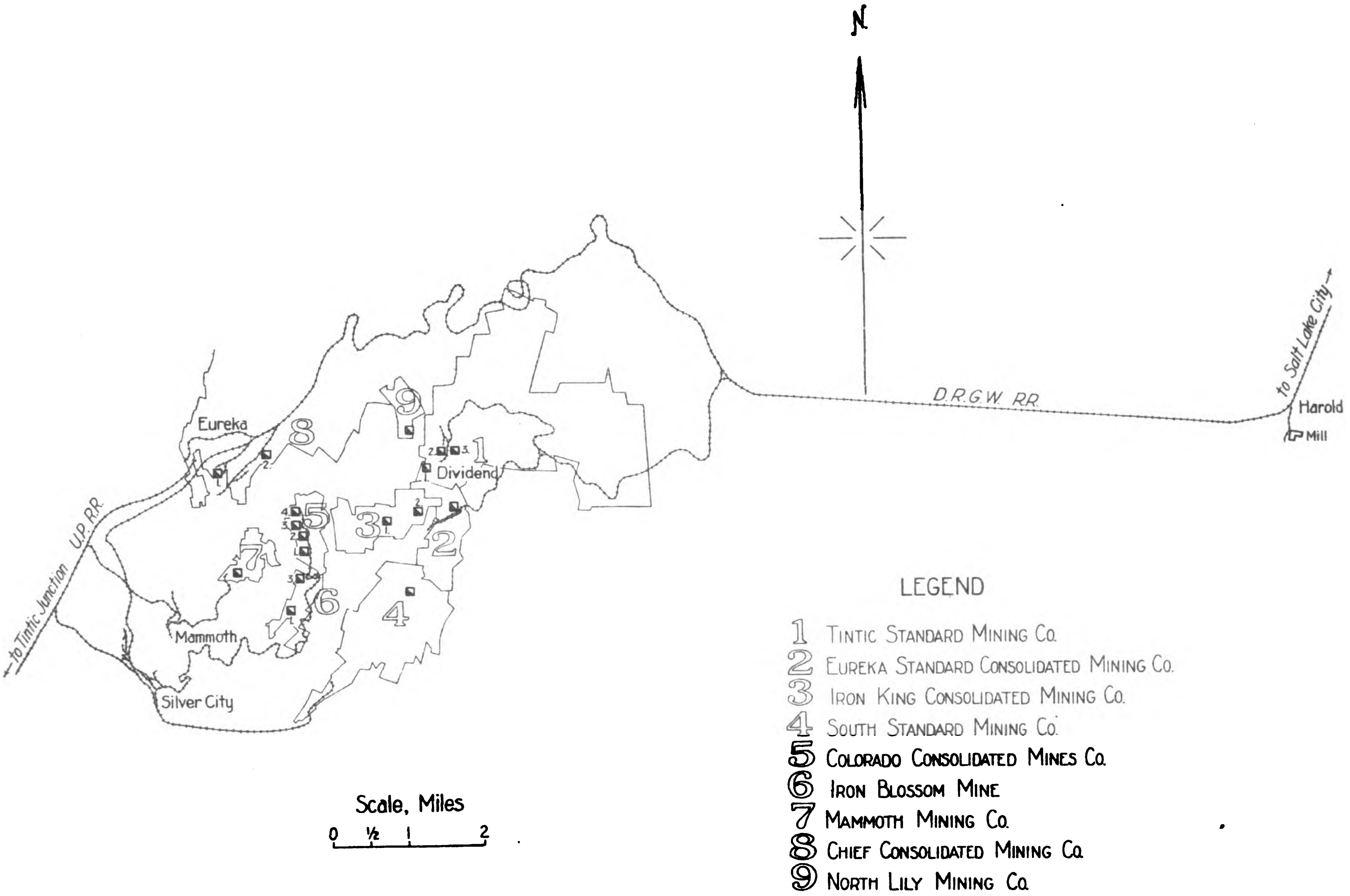


Figure 1. – *State of Utah, showing location of mining camps and smelters*



N
D.R.G.W. RR.
to Salt Lake City
Harold
Mill
Eureka
Dividend
Mammoth
Silver City
to Tintic Junction
U.P. RR.
1
2
3
4
5
6
7
8
9
Scale, Miles
0
½
1
2
LEGEND
1 Tintic Standard Mining Co.
2 Eureka Standard Consolidated Mining Co.
3 Iron King Consolidated Mining Co.
4 South Standard Mining Co.
5 Colorado Consolidated Mines Co.
6 Iron Blossom Mine
7 Mammoth Mining Co.
8 Chief Consolidated Mining Co.
9 North Lily Mining Co.

Figure 2.— Property map of the Tintic district

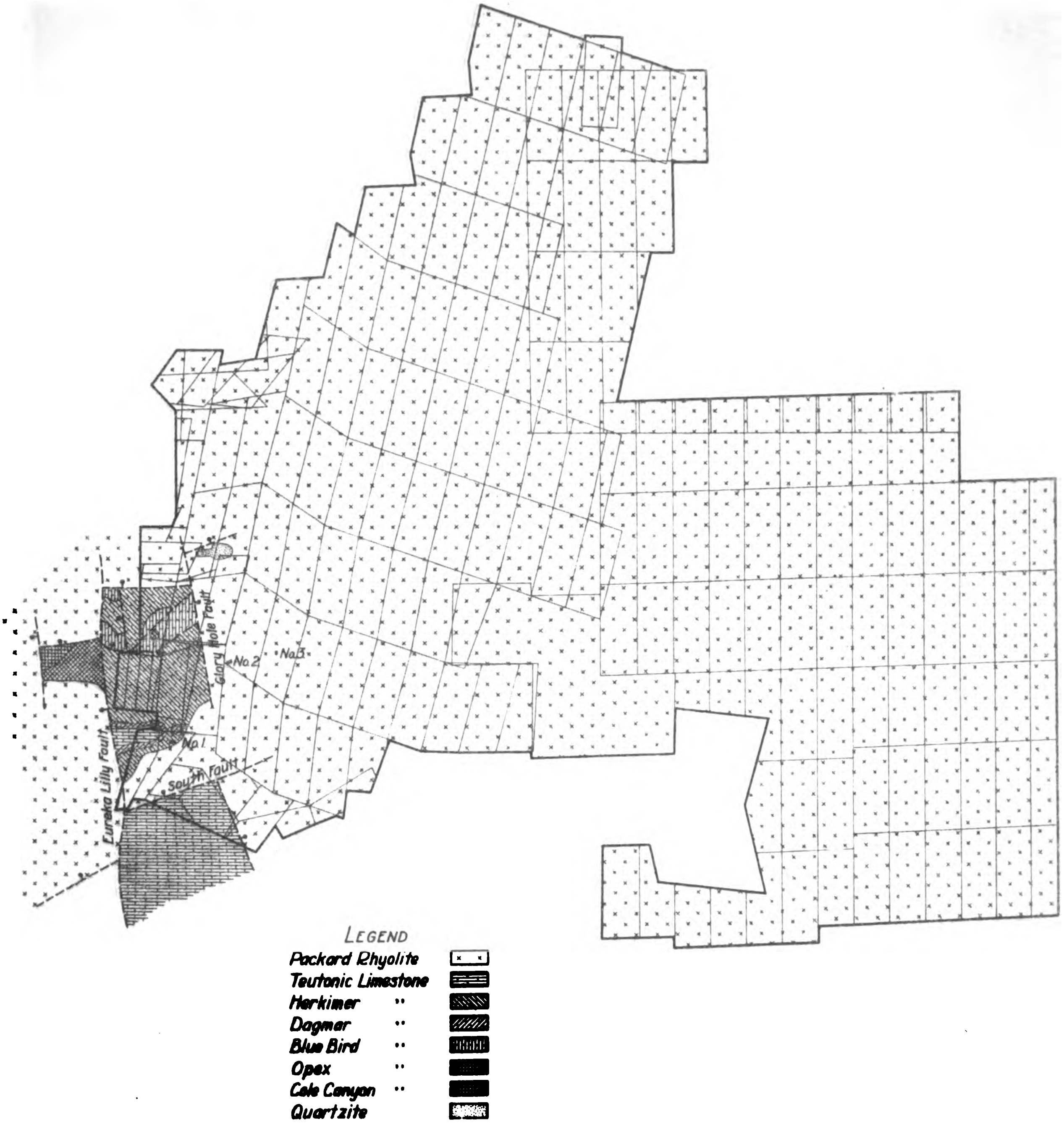


Figure 3. – Surface geology, Tintic Standard mine

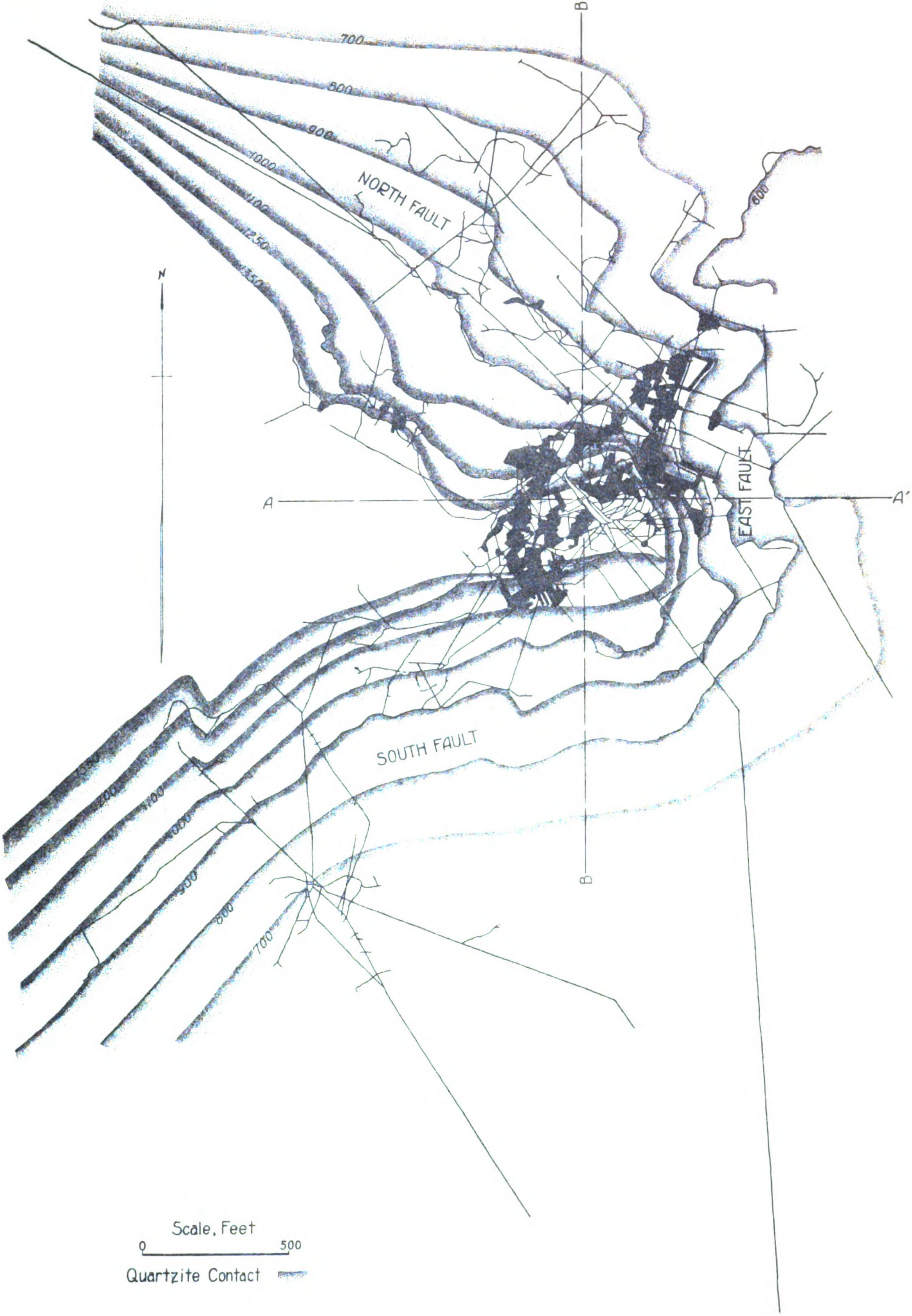
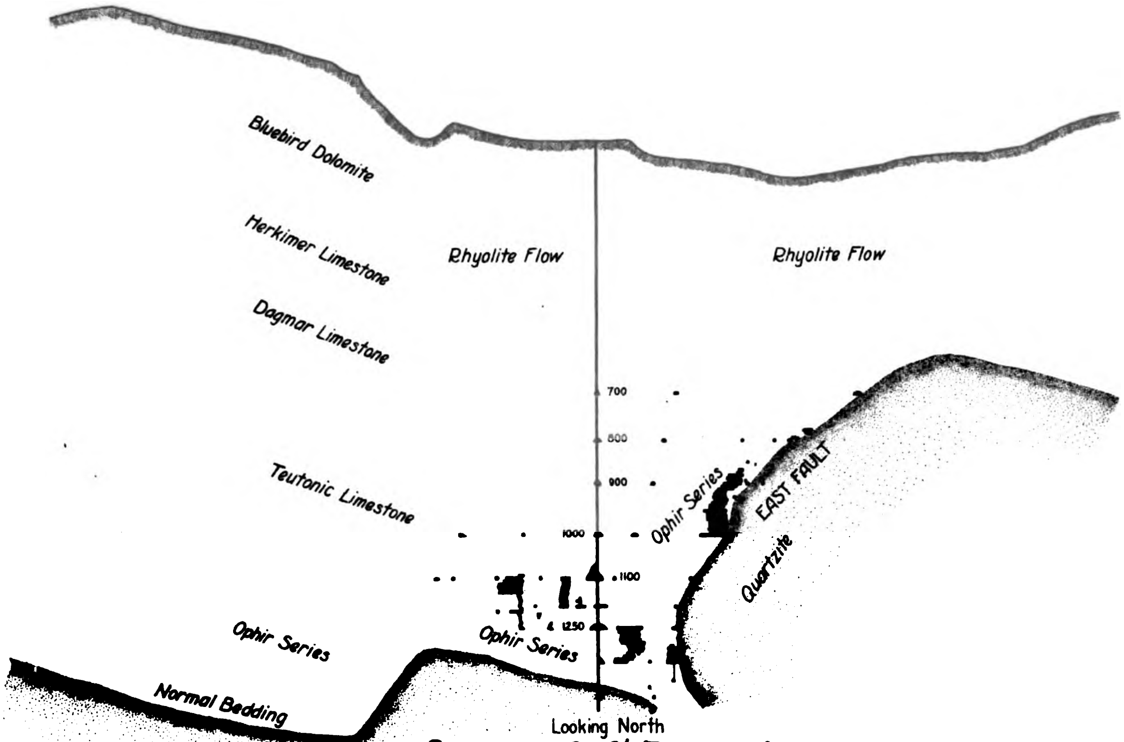


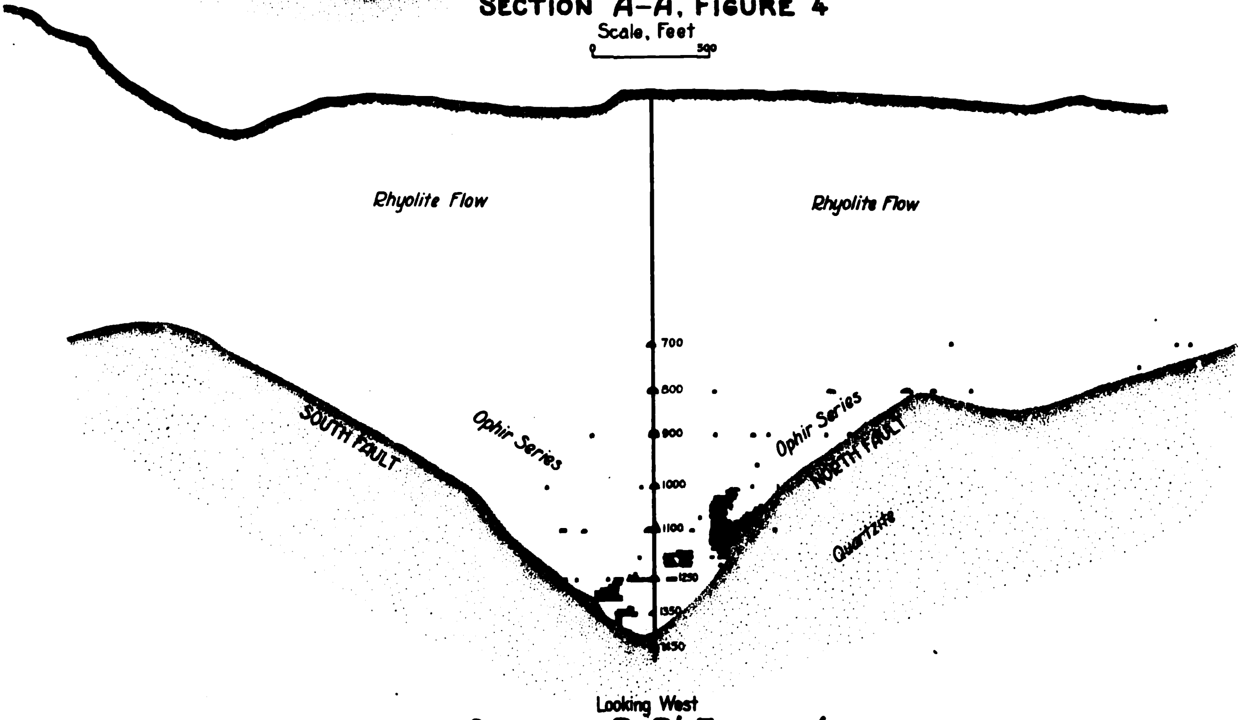
Figure 4. – Composite of Tintic Standard mine



SECTION A-A', FIGURE 4

Scale, Feet

0 500



SECTION B-B', FIGURE 4

Scale, Feet

0 500

Figure 5.—Sections through Tintic Standard mine

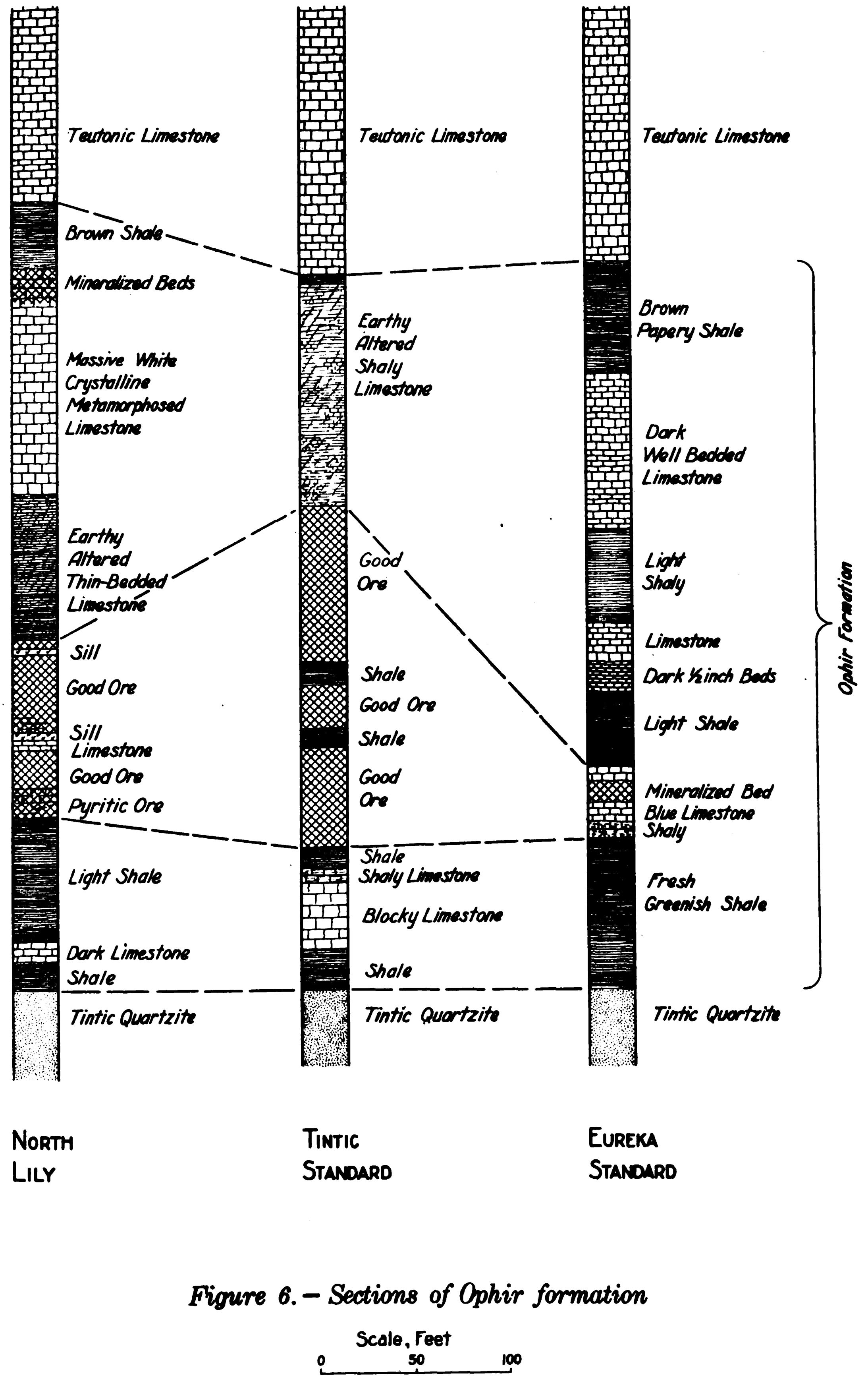


Figure 6.— Sections of Ophir formation

Tertiary time and affected the whole intermountain region. During this period the Tintic district was covered with many successive flows of lava, consisting mostly of rhyolite porphyry with some andesite and latites. The crowning event of this period was the intruding into the south end of the district near Diamond of a huge monzonite stock, which was forced up through the sedimentaries. Along with the great intrusion, or later but from the same source, came the mineral-bearing solutions that flowed out readily into the loose open channels already created in the soluble limestones, thereby making the replacement ore bodies from which many millions of dollars worth of metals have been produced.

The Tintic Standard mine is located on the east limb of the great Tintic syncline mentioned above. It comes very low in the general geological pile of Tintic, with the exception of surface flows being entirely in the Cambrian quartzites, limestones, and shales. Some 5,000 feet of this Cambrian quartzite underlies the shale and limestone series which produce the ore bodies of this mine. The surface geology is shown in Figure 3.

During the general seismic activity following the period of deposition of the sedimentaries three major premineral normal faults were formed in the Tintic Standard property. The first, called the South Fault, strikes about N. 80° E. and dips northerly about 45°, the down throw block being on the north. The second, called the North Fault, strikes northwesterly and southeasterly and dips southwesterly about 45°, the down throw block being on the south side. Thus, these two faults, dipping toward each other, form a V-shaped trough narrow on the east end and wider to the west with the down-throw block between them. A third, called the East Fault, later in occurrence than the first two, striking about N. 30° E. and dipping 70° westerly with the down-throw block on the west side, cuts this trough near its east or narrow end (see figs. 4 and 5.) On the up-throw side of each of these faults, or surrounding the trough, the top of the normal quartzite bedding is at about the 600 level of the mine; while on the down-throw sides, or within the trough, it is at about the 1,400 level, giving for all three faults a vertical throw of approximately 800 feet. The sedimentary beddings within the trough normally strike northerly and southerly and dip easterly 20 to 40°.

There then existed, prior to the deposition of mineral, a trough bounded on two sides and one end by major faults with faulted faces of quartzite from the 600 level to the 1,400 level, and with a quartzite bottom made up of normal beds dipping easterly so that the trough is deep and narrow at the east end and wider and more shallow to the west.

Figure 6 shows a section of the limestones and shales in the trough and overlying the quartzite which are called the Ophir series and which have contained practically all of the Tintic Standard ore bodies found to date. This series is composed of alternate lime and shale beds as shown, capped by a thick shale bed which seems to have had some damming effect on the ascending mineral-bearing solutions.

Extending across the deep, narrow east end of the trough and striking north 30 to 50° east are several vertical fissure zones through which mineral-bearing solutions came from some unknown source at depth and, upon reaching the much broken and shattered soluble limestone beds, found easy access and many points of attack for replacement. The line of least resistance for these ascending solutions was along the major faults, so that the large replacement ore bodies are found in the Ophir limes adjacent to faulted faces of the quartzite.

The final geological event that influenced the molding of these ore bodies of the Tintic Standard mine was the occurrence of a fourth system of normal faulting, which is mostly postmineral. This system strikes about N. 20° W. and dips steeply to the southwest, forming a succession of step faults which have cut the ore bodies and raised the northeast segments successively higher as shown in Figure 7, an ideal section through the ore zones at right angles to the postmineral fault system last mentioned.

PHYSICAL CHARACTERISTICS OF ORE AND INCLOSING ROCKS

As stated before, the major ore bodies of the Tintic Standard mine are found in the Ophir limestone on or near its contact with faulted faces of Cambrian quartzite, a firm, hard, massive rock. Along these fault faces, both premineral and postmineral faulting has been extensive.

The principal revenue of the mine is from lead-silver ores, with a quartz-barite gangue. The average silica content of clean, well-sorted lead ore is 49 per cent. The following analysis is typical of the average of lead-ore shipments since the opening of the mine:

Lead.............	25.04	per cent
Copper...........	.31	per cent
Silver...........	30.29	ounces
Gold.............	.039	ounce
Silica...........	49.03	per cent
Iron.............	7.18	per cent

Occurring with lead ore in large masses and representing an earlier phase of mineralization than the lead is a siliceous ore of the following average content:

Lead.............	4.85	per cent
Copper...........	.36	per cent
Silver...........	17.24	ounces
Gold.............	.037	ounce
Silica...........	66.96	per cent
Iron.............	11.50	per cent

Both of these ores are completely oxidized above the 1,000 level, while below that level the sulphide content of the ore gradually increases, until on the 1,450 level the ores are practically all sulphide. Extensive postmineral faulting has brecciated a considerable part of this ore into fine sandy material, intermixed with masses of heavy lead ore. The hanging-wall rocks are the shales and limestones. The hanging-wall shales have been altered by mineralizing solutions to a soft, gray, clayey material. The limestone areas over the ore bodies

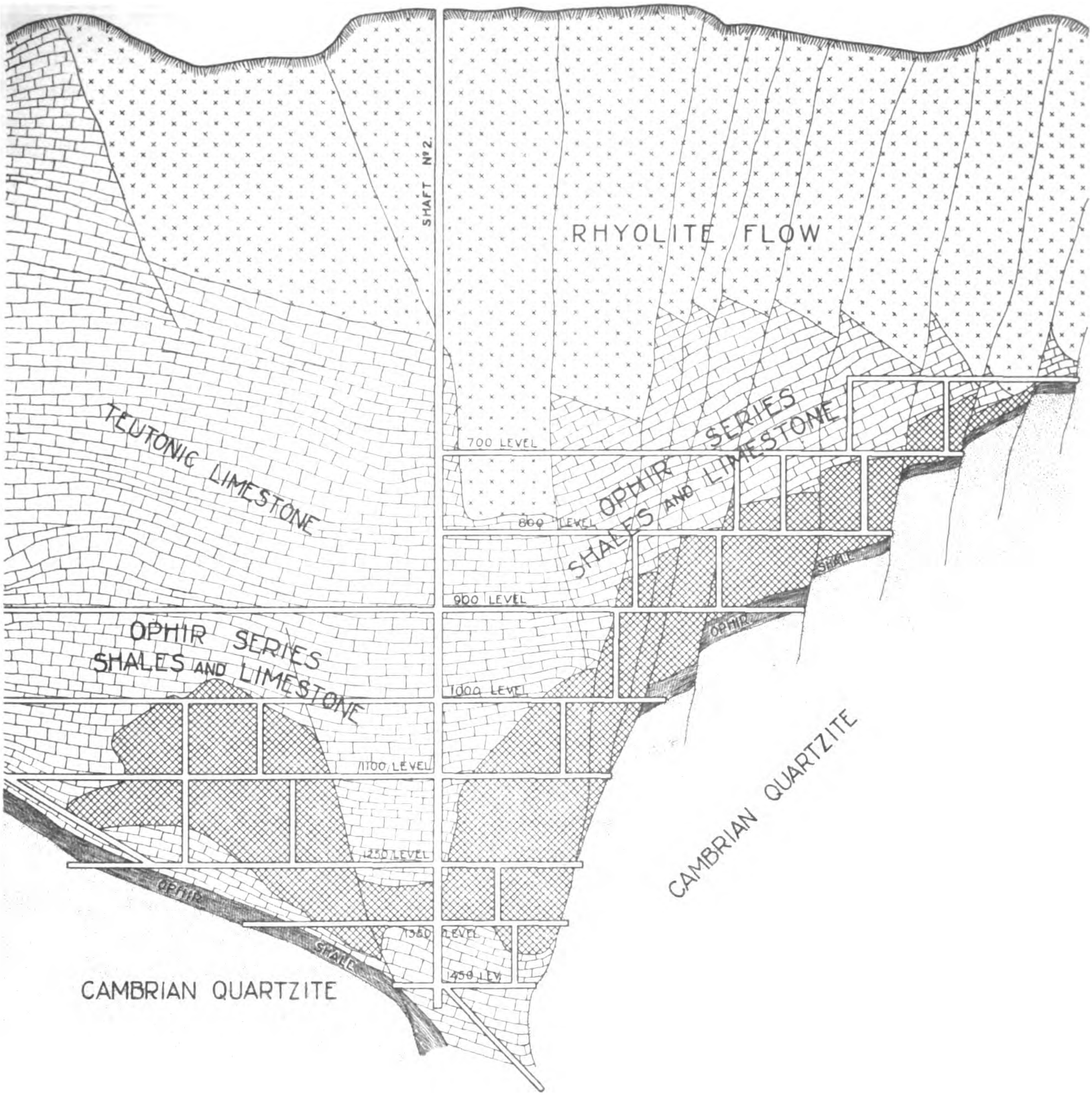


Figure 7. – Ideal section on line N. 45° 0′ E. through No. 2 shaft

Scale, Feet

0 300

are altered by the introduction of silica and magnesium silicates. These altered limestone areas are brecciated and have the appearance of the collapsed breccia filling in limestone caves.

The individual ore bodies, when adjacent to the quartzite fault faces, have steep dips ranging from 45° to vertical; when distant from the quartzite fault faces, they are tabular replacements of flat-lying limestone beds. The tabular commercial ore bodies range in height from 6 to 200 feet.

PRODUCTION

The average annual production of the Tintic Standard mine for five years has been 121,265.995 tons. The variation in output is influenced largely by variation in metal prices. A large tonnage of siliceous ore of rather low grade, existing in the mine, can be moved at a profit only when metal prices are favorable; it is in this class of ore that curtailment is made necessary by a decline in the price of silver below 60 cents per ounce and lead below 6-1/2 cents per pound. The highest year's production was in 1926, with 132,165.400 dry tons of ore containing 45,689,463.14 pounds of lead and 3,433,264.02 ounces of silver. The total production from discovery to December 31, 1929, was 1,328,787.82 dry tons of ore, containing 361,553,421.5 pounds of lead, 35,037,932.43 ounces of silver, 10,181,872.95 pounds of copper, and 47,692.78 ounces of gold.

METHODS OF PROSPECTING AND EXPLORATION

All prospecting and exploration work is based on a thorough study of the geology. Each successive step in the progress of development is a result of those preceding. Preliminary knowledge was gained from careful mapping of the geology of the old workings; at present all new workings are closely observed, and the geology is recorded by the geological department.

Exploration is done through two shafts sunk to the Ophir formation and by drifts from the shafts into the ore measures in those areas where these measures are intersected by the fissure system (see figs. 4 and 7).

As structural geological features determine the ore bodies in this district shafts, raises, winzes, drifts, and crosscuts are used nearly exclusively for prospecting. A very small amount of diamond drilling has been done, and at present churn drilling is being successfully used to determine the depth of rhyolite flow, the thickness of underlying limestone, and the top of the quartzite in outlying sections of the property.

Ore bodies are found by drifting in favorable horizons on the quartzite-limestone or shale-limestone contacts. These same prospect drifts are used to extract the ore found, until costs of upkeep require others as explained later. Outlines of the ore bodies are clearly defined by structural features and are developed only in the process of mining. The hanging-wall is drilled at short intervals during mining, and the foot wall is prospected by extraction drifts.

METHODS OF SAMPLING AND ESTIMATION OF TONNAGES AND VALUES

All of the material hoisted, both ore and waste, is sampled three different times; waste which is used for filling is sampled twice.

The first samples taken are face samples. These are taken by especially trained men and under direct supervision of the engineering department. A portion of the mine is assigned to a sampler; it is his duty to sample daily and to record the assays of every operating face in that portion of the mine. Samples are taken in drifts and raises and every square-set face in the stopes opposite both the cap and the girt. These samples are grooves made with a hand pick at right angles to the dip of the vein and weigh approximately 5 pounds. Where a face is made up of two or more streaks of different hardness or mineral content each streak is measured and sampled separately. These samples are sent to the assay office by 10:00 o'clock a.m., and the determination is completed by noon. Each shift boss is furnished a complete assay sheet, showing all the samples taken, and these sample sheets are used by him in determining the breaking and classifying of ore in his section of the mine. In addition, the assays are copied on small cardboard tags and nailed on the cap over the face from which they were taken. This enables the miner himself to have full knowledge of the ores in which he is working; a most important item when sorting is to be done. Assay maps of each floor in the stope, raise, or winze and continuations of drifts and crosscuts are brought up to date twice a week by the samplers. These assay maps contain the dates of mining and details of lacing and filling for each set along with the face assays (see fig. 8). On this sheet colors are used to indicate the month in which ore was mined. Average values so obtained are checked against actual smelter returns. These records are used in the estimation of ore reserves and by the management for the control of metal shipments.

The second sampling is a hand-grab sample taken by the trammers as they draw the chutes. These are referred to as box samples; one of which is taken from each section of the stope and from each drift. Box samples act as a check on the face samples, and in this way each round of waste broken is resampled before a gob is entered.

The third sampling consists of taking a "composite" grab sample from the cars on the surface by the top-carmen of each grade of ore and of the waste hoisted, as a final check before the ore enters the freight cars.

The grab samples taken by the shift bosses and top-carmen are used to determine the value of the ore produced that day. Reports made by the shift bosses of the number of cars from each section, which is checked by the top-carmen's report of cars of each grade of ore, are used to estimate the tonnage for the day. These figures are used in calculating the daily gross earnings.

Ore reserves are clearly defined by structural limits and computed by weighted assays taken from the assay maps.

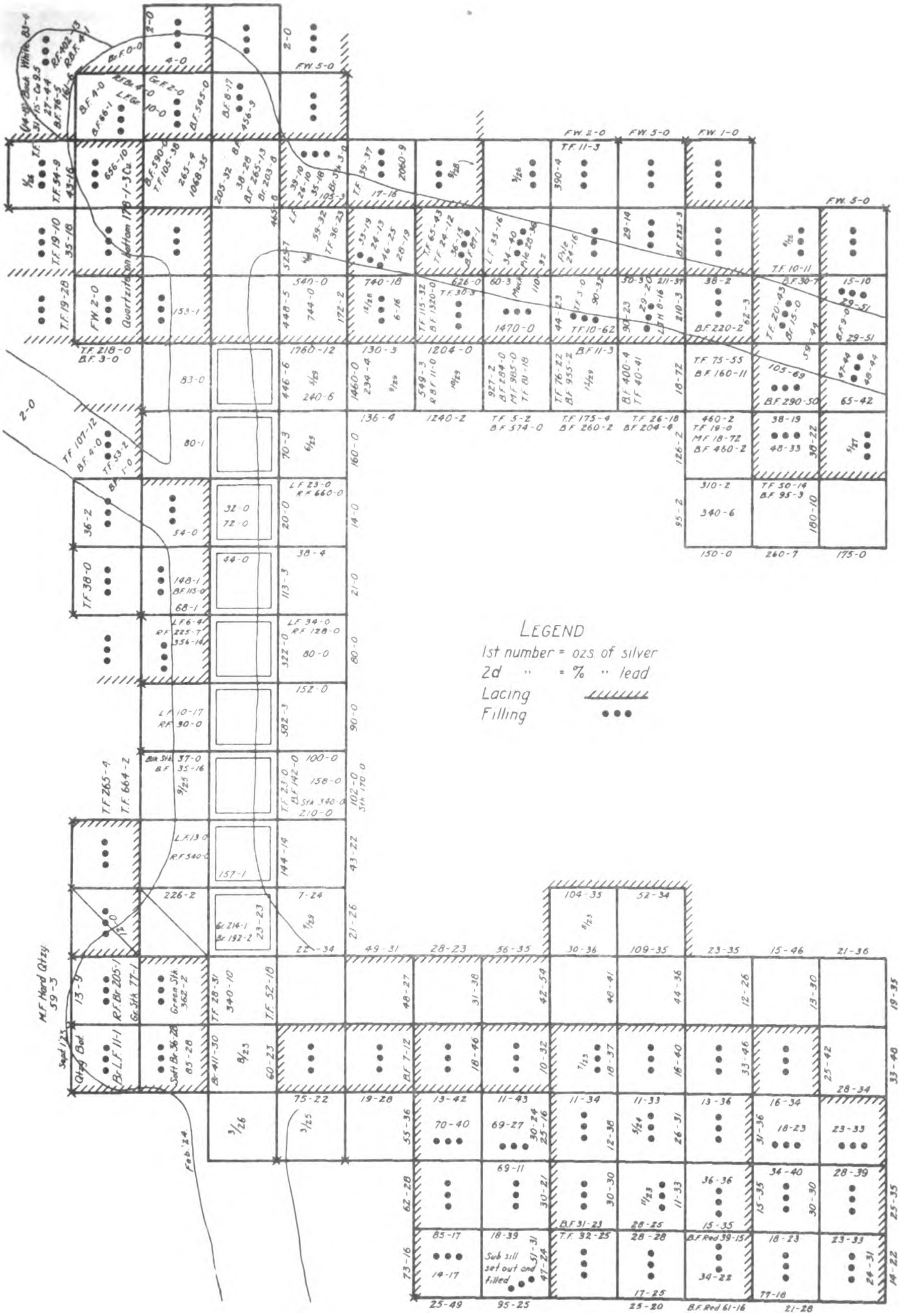


Figure 8.— Assay map of the floor of a stope

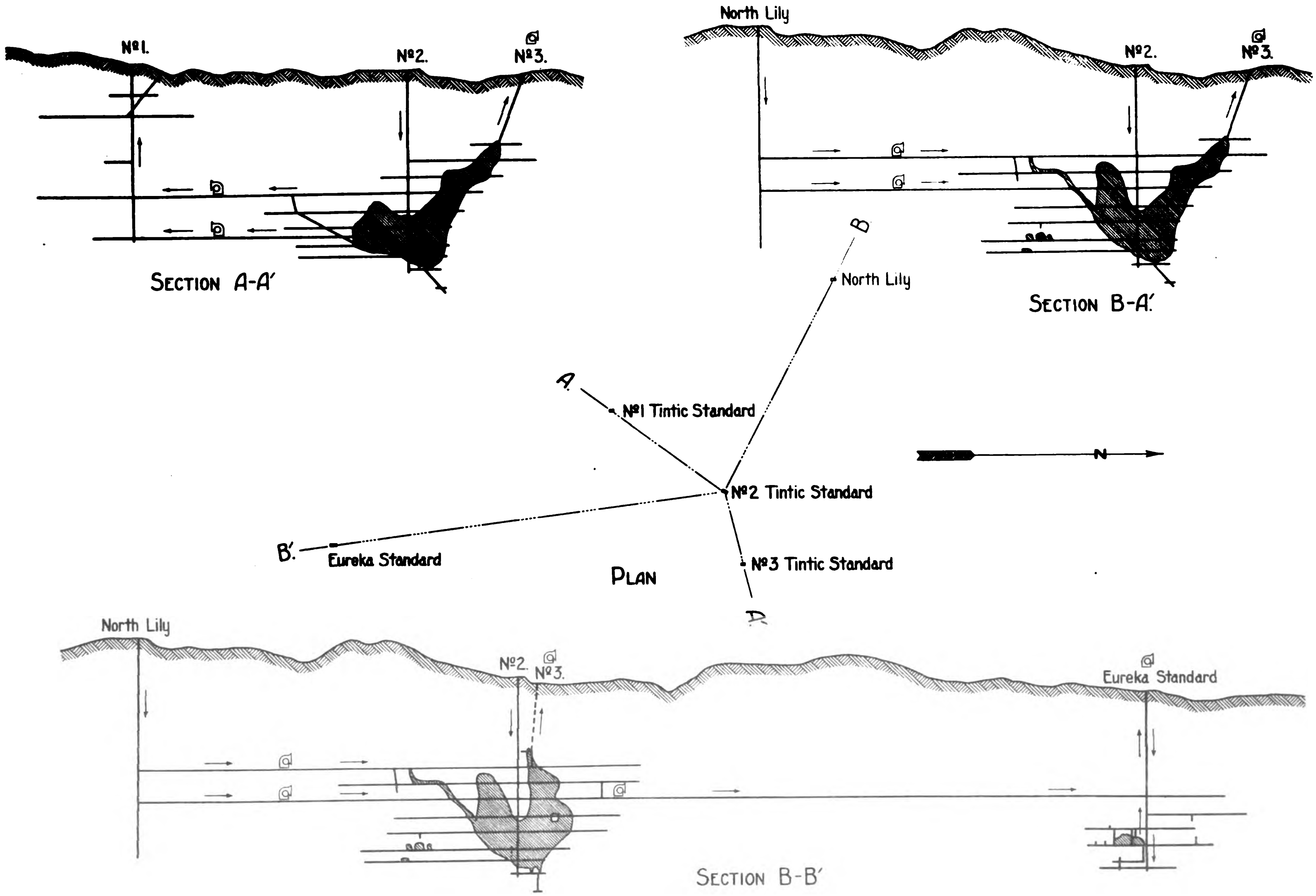


Figure 9.— Ventilation map

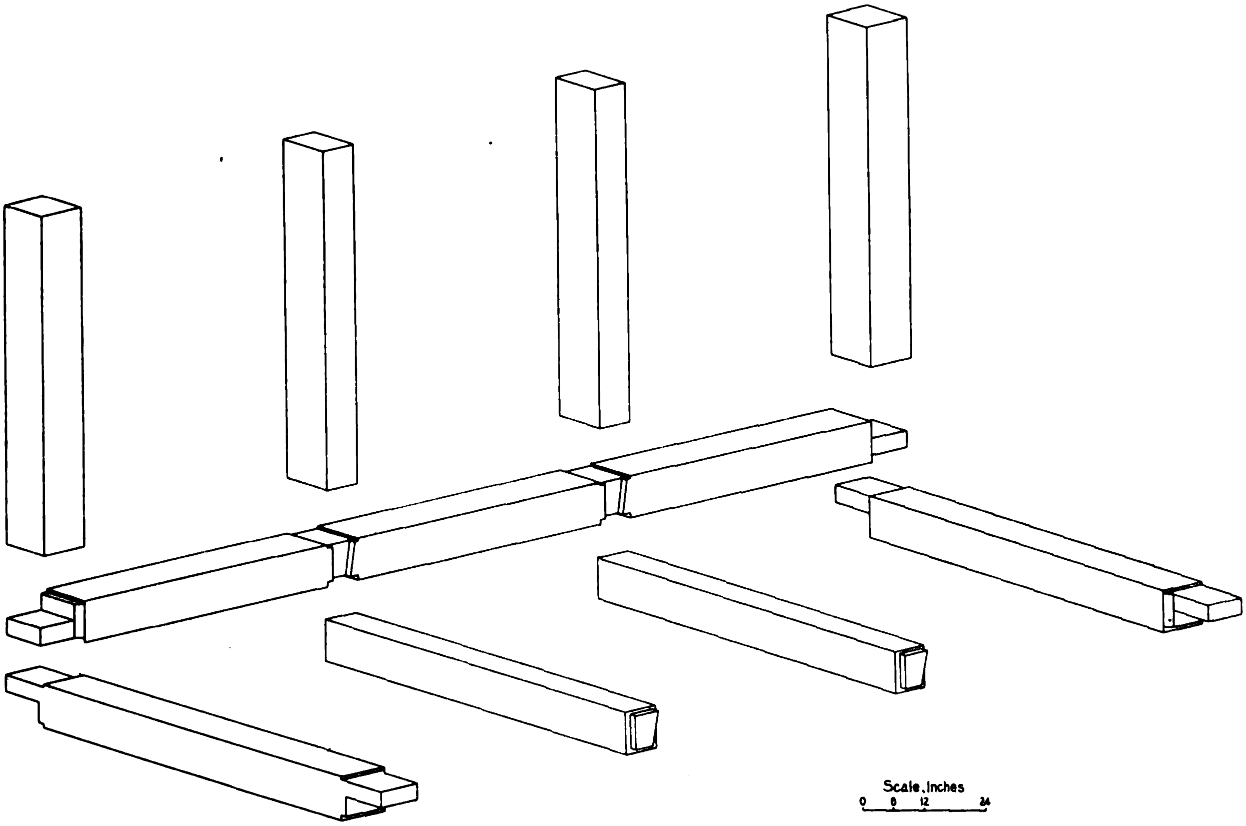


Figure 10. – Shaft set

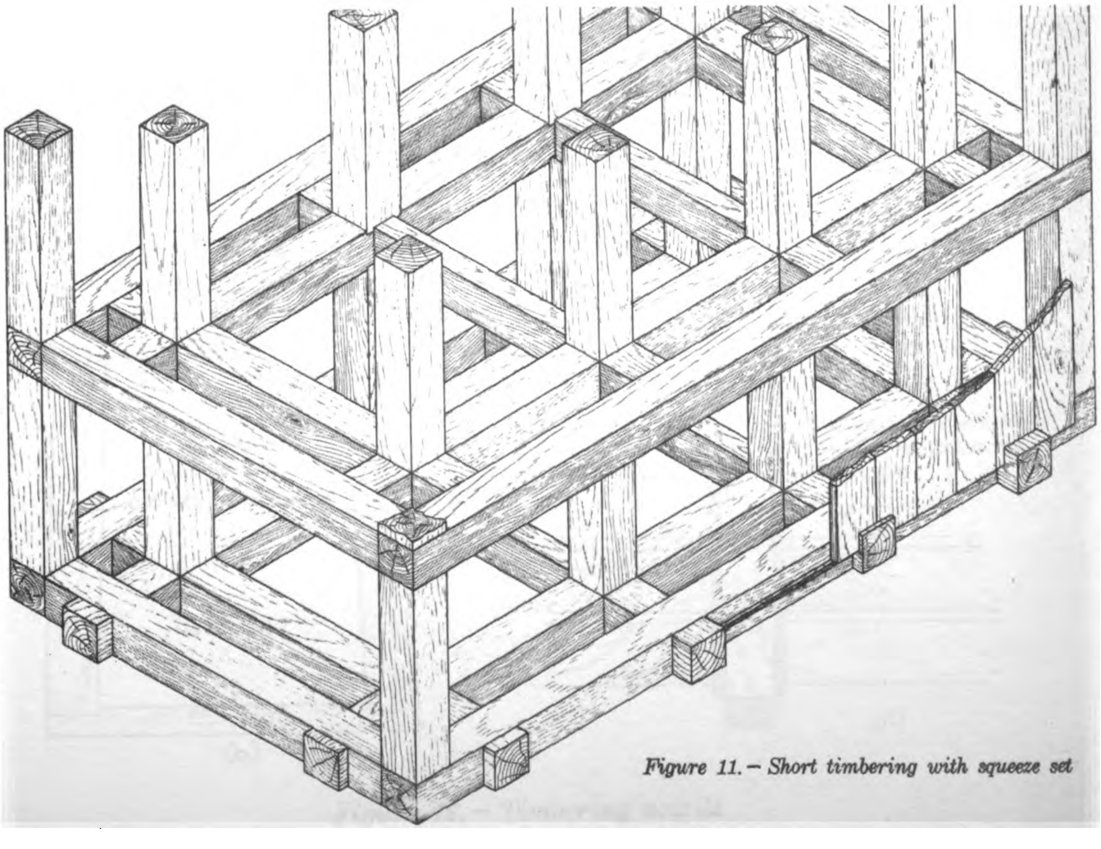
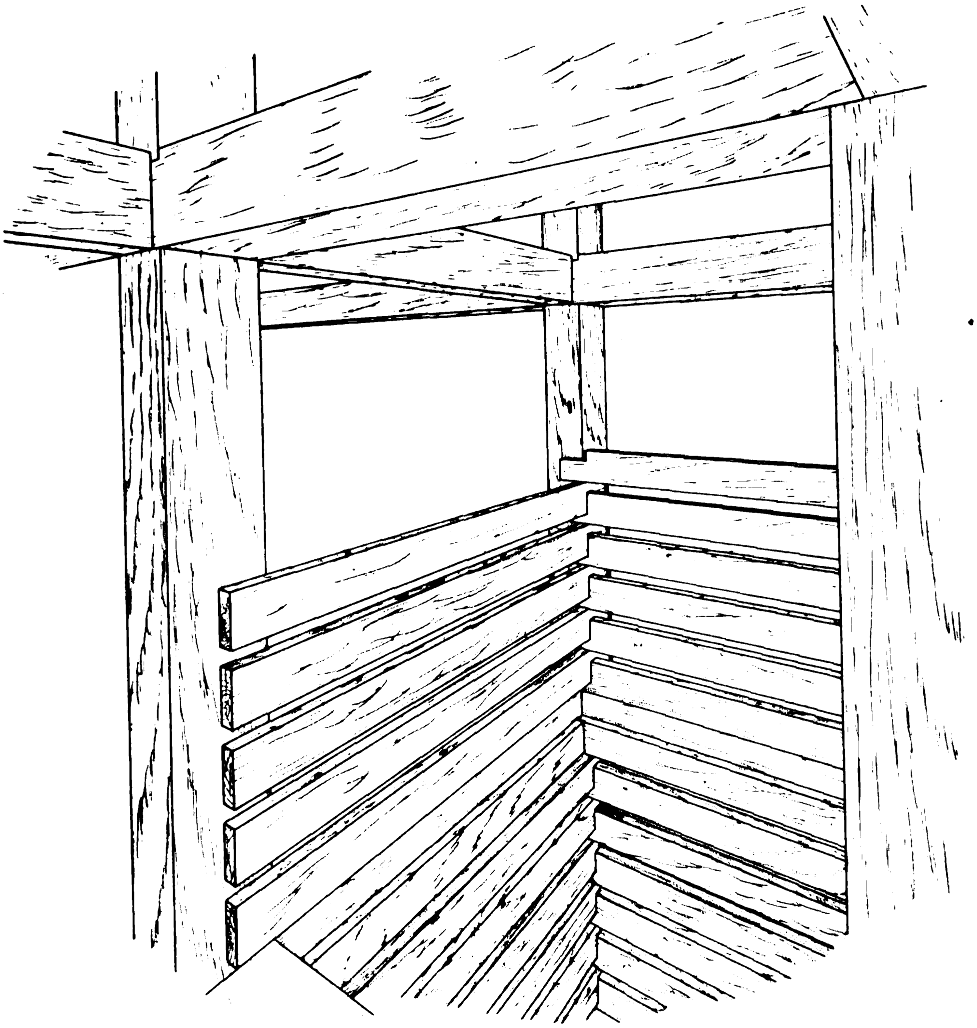
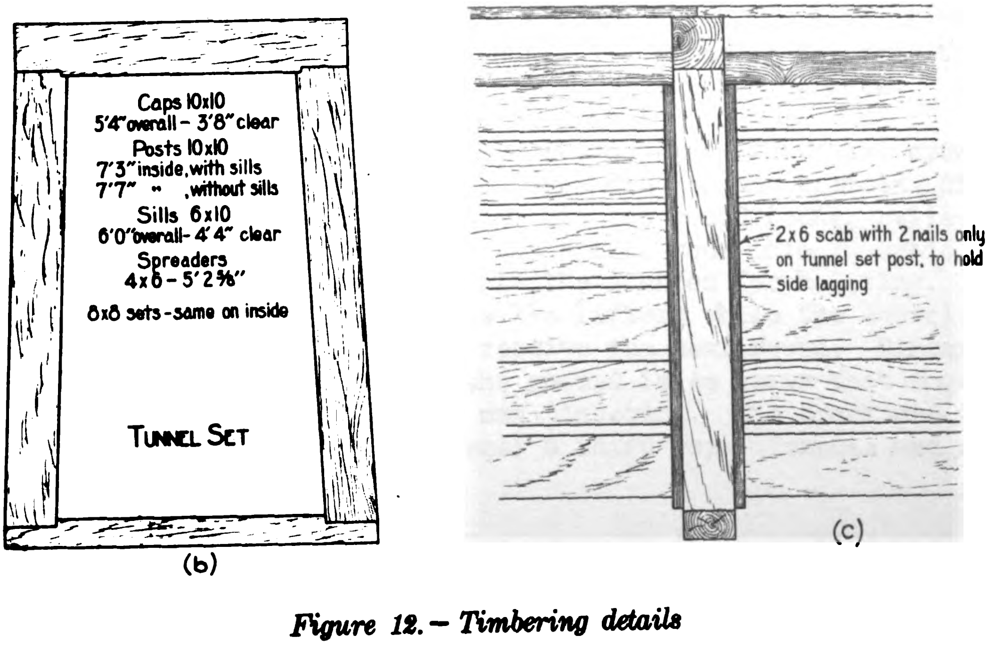


Figure 11. – Short timbering with squeeze set



CORNER LACING (a)



Caps 10x10
5'4" overall – 3'8" clear
Posts 10x10
7'3" inside, with sills
7'7" " , without sills
Sills 6x10
6'0" overall – 4'4" clear
Spreaders
4x6 – 5'2⅜"
8x8 sets – same on inside

TUNNEL SET

(b)

2x6 scab with 2 nails only on tunnel set post, to hold side lagging

(c)

Figure 12.– Timbering details

METHODS OF DEVELOPMENT AND MINING

Shafts

The mine is operated through three shafts, Nos. 1, 2, and 3 (see fig. 9, sec. A-A'). The No. 1 shaft was sunk first. From this a large amount of prospecting was done, and the ore was struck on the 1,300 level, corresponding to the 1,200 level of the present No. 2 shaft. Due to the distance from the ore and several transfers and to the hoat, the No. 2 shaft was sunk in 1916; it has been used ever since as the main shaft, through which all ore, men, and supplies are hoisted and lowered respectively.

No. 2 shaft has two hoisting compartments, each 4 feet 6 inches by 4 feet in the clear, and a manway compartment, 4 feet 6 inches by 2 feet, for air and water pipe, electric cable, and a ladderway. It would have been preferable to have had this latter compartment the size of the other two. It is timbered with 8 by 8 inch timber sets, spaced 6 feet apart. Framing for the sets is shown in Figure 10. The sets are tightly lagged on four sides with 2 by 12 inch by 6 foot timber, the lagging being placed vertically. In doing repair work in the lower levels the manway compartment has been increased to full size and the size of the timber changed to 10 by 10 inch; outside of the regular shaft set has been placed a 10 by 10 inch "squeeze" set with 10 inch blockings to the shaft set. In this case the "squeeze" set is lagged (see fig. 11).

No. 3 shaft an incline (the other two being vertical), was sunk for ventilation; No. 1 shaft has also been used for this purpose since being replaced as the hoisting shaft by No. 2.

Tunnels, drifts, and crosscuts

Drifts and crosscuts are 6 feet wide and 8 feet high. The back is arched for natural support where this is possible. In the most part, 10 by 10 inch tunnel sets are used with 6-foot spacing; the framing and size of the sets are shown in Figures 12 (b) and 12 (c). When crosscutting or drifting through ore the standard sill square set is used in place of the tunnel sets.

Drilling and shoveling are done simultaneously. The usual crew in a heading is a miner and one or two muckers, depending on the tramming distance. The miner, standing on the muck pile, puts in his top round (cuts, relievers, and back holes), while the shovelers load the broken rock into 18-cubic foot cars and tram it to a chute for filling or to the station for hoisting. When the pile has been removed, the miner drills his lifters while the shovelers lay a platform of 2 by 12 inch lagging to receive the next round. Timbering is usually done by a separate crew; when the ground is so heavy that immediate timbering is necessary the miner does his own timbering. The track and the air and water pipe are advanced by special crews; a third crew advances and cares for the ventilation pipe.

A draw-cut round is usually drilled with a jackhammer using 7/8-inch hollow hexagon steel with the standard double taper (14^0 and 5^0) cross bit. Starters are 3 feet long with a 2-inch gage; changes are of 1/8-inch decrease in gage for an increase in length of the steel of one foot. If the ground is too hard for a jackhammer the miner uses a Leyner machine mounted on a horizontal bar. Inch and a quarter hollow round steel with the double taper (14^0 and 5^0) cross bit is used with the Leyners. Starters are 3 feet long with 2-1/2 inch gage; changes are of 1/4-inch decrease in gage for an increase in length of 1-1/2 feet. The depth of the round is usually 4 feet. No dry drilling is done in the mine. Each miner loads and fires his own round. Twenty-five per cent strength gelatin dynamite is used in the softer ground and 40 per cent strength in the harder.

Raises

All raises are timbered with standard square sets described later and completely lagged with 2 by 12 inch timber. They are two sets wide, one of which is used for a manway, timberway, and air, water, and ventilation pipe; the other set is for a chute. These raises are vertical and rarely exceed 120 feet.

Two men always work in a raise - a miner and a mucker. The chute compartment is lined to within two sets of the top. On top of the chute 8 by 8 inch timber spaced 1 foot apart acts as a grizzly. A slide is built over the manway compartment, or it is lagged tightly over with 8 by 8 inch timber. All timber and supplies are hoisted by hand with a rope and pulley. In bad ground only one set at a time is advanced, but when possible room for both sets is shot at the same time. The miner and mucker timber as soon as room is made for an additional set. Raise rounds consist of a V-cut in the center, the other holes expanding the V to the outside with each succession of shots. Stoper machines are used with 1-inch quarter-octagon hollow drill steel. Starters are 3 feet long with a 2-inch gage; change in gage is 1/8-inch for an increase in length of one foot. Special crews care for air, water, and ventilation pipe as in the drifts. Blasting is also similar; fuse varying from 6 to 8 feet in length is used.

Stoping Methods

The importance of selecting a right method of mining for any particular ore body must not be underrated. To make that selection a great variety of facts must be assembled, and their relation to profits to be won must be understood. Of major importance before a method can be selected, are the following items:

1. Characteristics of the ore body.
    - A. Size of ore body.
    - B. Shape of ore body.
    - C. Character of ore.
    - D. Character of wall rock.
2. Grade of ore to be mined.
3. Metallurgical treatment to which ore will be submitted.

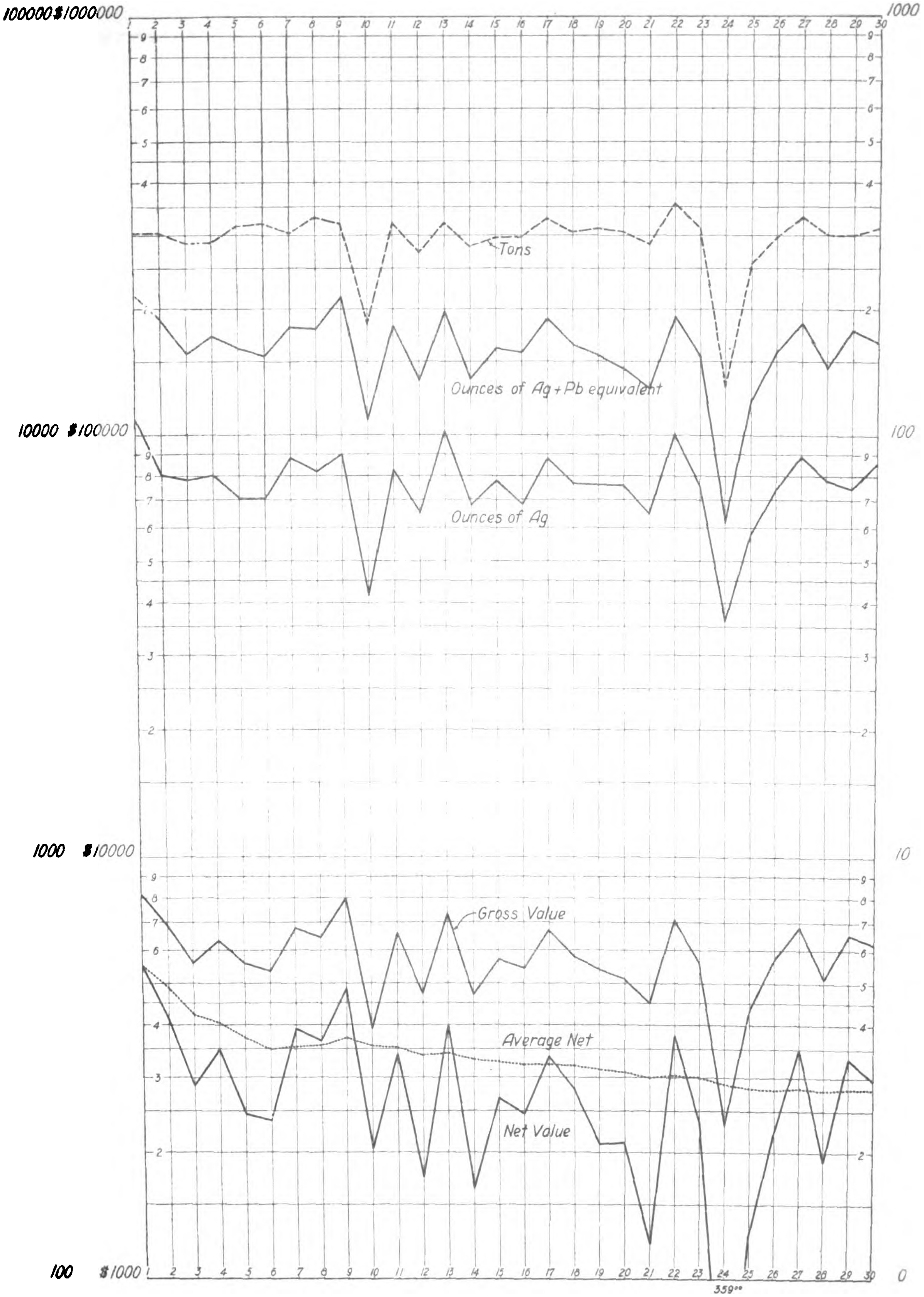


Figure 13. – Production curves for month of November, 1929

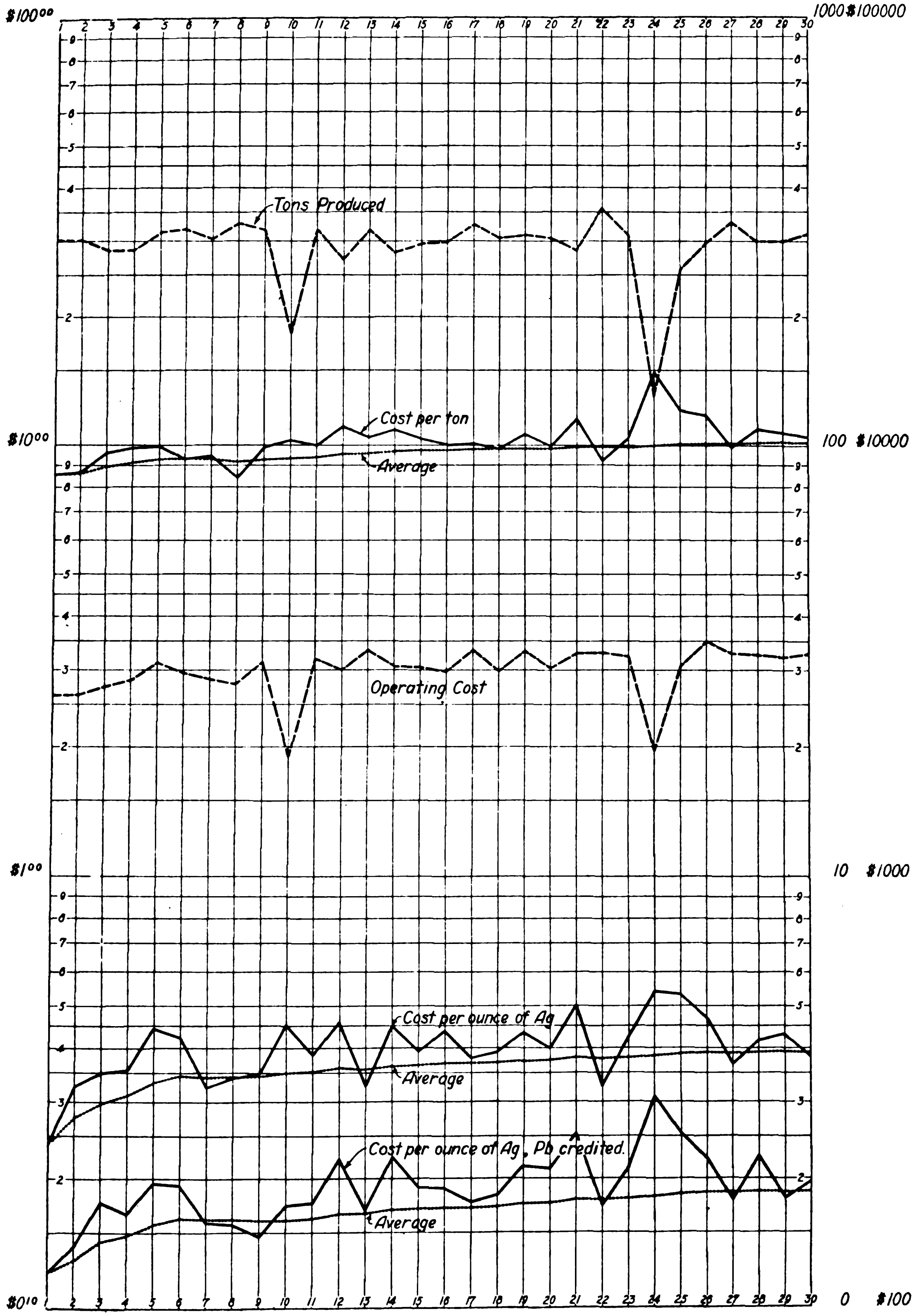


Figure 14. – Cost curves for month of November, 1929

4. Availability of labor.
5. Availability of supplies.

With these five items well in mind items of less importance must be considered and the method of mining modified to give the best results. The majority of ore bodies do not cease to carry recoverable minerals at a definite limit, but the values diminish gradually at the stope boundaries. In general, the lower the cost per unit of producing the metal can be forced and the higher the metallurgical recoveries can be established the larger the ore body.

At the Tintic Standard mine the rapid change in the ore from lead ore to dry silver ore and to low-grade quartz makes stope control essential. As stated before, all new faces are sampled. Mining ore of this character imposes the duty of knowing when a stope ceases to be profitable; an estimate is made daily of the total mine profit. Individual daily records of each stope showing grade of ore, quantity of ore, and the cost of mining the ore, are charted (see figs. 13 and 14), and from these data mine control can be exercised.

Production charts show: Tons of ore, ounces of silver, and ounces of silver with lead credited as silver. These curves are plotted each day on logarithmic paper, their relationships studied, and any unusual occurrence noted and corrected. The same type of charts is made for costs and their relationship.

The stoping methods first used at this mine were based upon following the ore. Both open stopes and square sets were used, depending upon the height of the ore body. The timbers for the square sets were light. Eight by eight inch timber was used, and its principal purpose was to furnish a scaffold upon which to work. No effort was made to mine the entire ore body systematically or to provide support for the hanging wall if extensive mining operations should be developed, hence no provision was made for filling. Only high-grade, direct-smelting ore was commercial, and stopes were developed as irregularly shaped cavities in the larger mass of low-grade ore.

At present, with more complete knowledge as to the extent of commercial ore bodies in the mine, this policy seems unwarranted. However, at that time it was assumed the ore bodies would be similar to those found in the Iron Blossom and Colorado Mines, irregular replacements in limestone beds with small cross section. At the Tintic Standard, ore sufficiently valuable to ship alternated with masses of siliceous silver ore, much greater in tonnage than the higher-grade ores. These siliceous ores became commercial by the building of a mill, and later by a reduction in treatment rates by the smelting company, after the company found that the ores were needed in its operation.

In 1921 it was found necessary to fill the mined areas to preserve the unmined ore. However, filling did not proceed fast enough, as development-work waste was insufficient and no auxiliary waste was available. Severe caving within and adjacent to the mined ground resulted.

During this period a shrinkage-stoping method was tried and found unsuitable to the conditions. A variation of square-set stoping called the Miller or Mitchell system was tested and discarded. It did not furnish sufficient support to hold the soft hanging wall, as the pillars of ore would crumble.

After the failure of these methods it was decided to use regular square-set stoping and back fill. This method is still in use. Before systematic square setting could be done it was necessary to fill the open stopes then existing to permit safe mining near them.

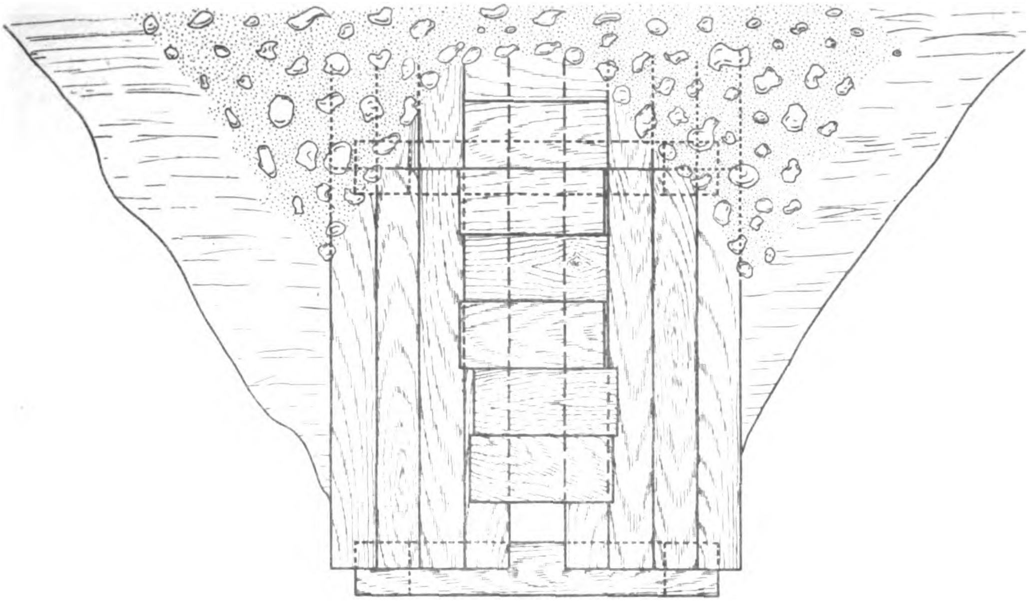
An open cut on surface was used to supply additional waste. An adit was driven into the side of a hill about 200 feet from the shaft. Ground above this adit was broken, beginning at the entrance, and loaded into cars by the method shown in Figure 15. Waste from the surface was lowered in cars, on the cages, to the different levels. In this way filling, along with waste from exploration and development work, was disposed of fairly well after about a year of hard work. All labor connected with mining, loading, and delivery of waste from the open cut to the collar of shaft was contracted at 20 cents per ton.

The general plan of stoping for the past 8 years has been to drive a vertical square-set raise from the footwall at the level the stope is to be started, to the level above (see A in fig. 16). The raise is the place from which a stope is started and through which waste is dumped from the level above when filling is wanted. For every stope (not section) started it is necessary to have, at least, one raise through to the level above for waste filling.

The first section of a stope started at the raise is cut two or three sets wide (depending on the kind of ground) and extending from the raise to the hanging wall. This section is mined floor by floor as high as safety will permit, then filled, leaving manways on the outside of the section at convenient places (see B in fig. 16). These manways afford a place of attack when the next section is started besides giving access to any floor above without having to raise; as three sides are in solid material they are safe and less expensive to maintain.

The original section is completed to the hanging wall or mined 8 floors high and filled before a new section two sets wide is started from the sill on one side. While the second section is being filled a third is started on the other side of the original section. By alternating from one side to the other a regular tonnage of ore can be maintained.

After a section has been mined and filling is needed it is thoroughly cleaned of broken ore. The floor lagging above the sill floor is taken up and nailed on the inside of the posts against the unmined ore. These gob laggings are installed 2 inches apart. A tight floor is placed on sills, consisting of one layer of 2 inch lagging, covered with a foot or so of refuse timber which forms a mat. The mat should not exceed 1 foot if excessive settling is to be avoided.


PLAN
of
LOADING TRAP

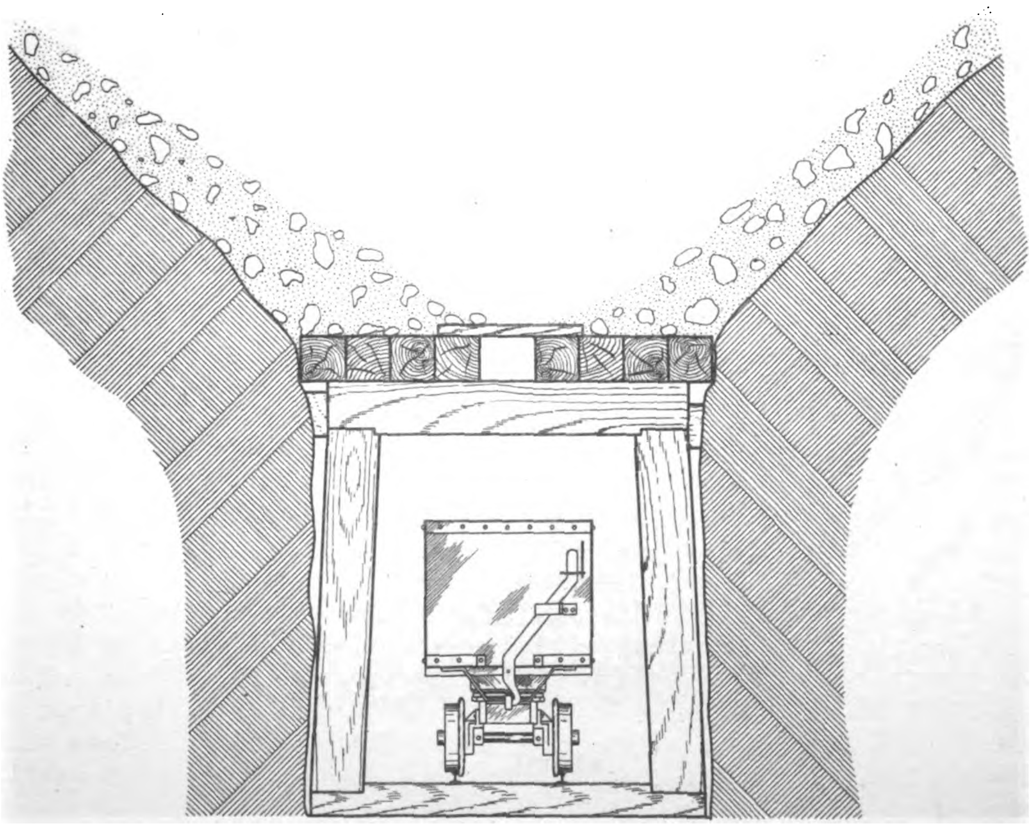
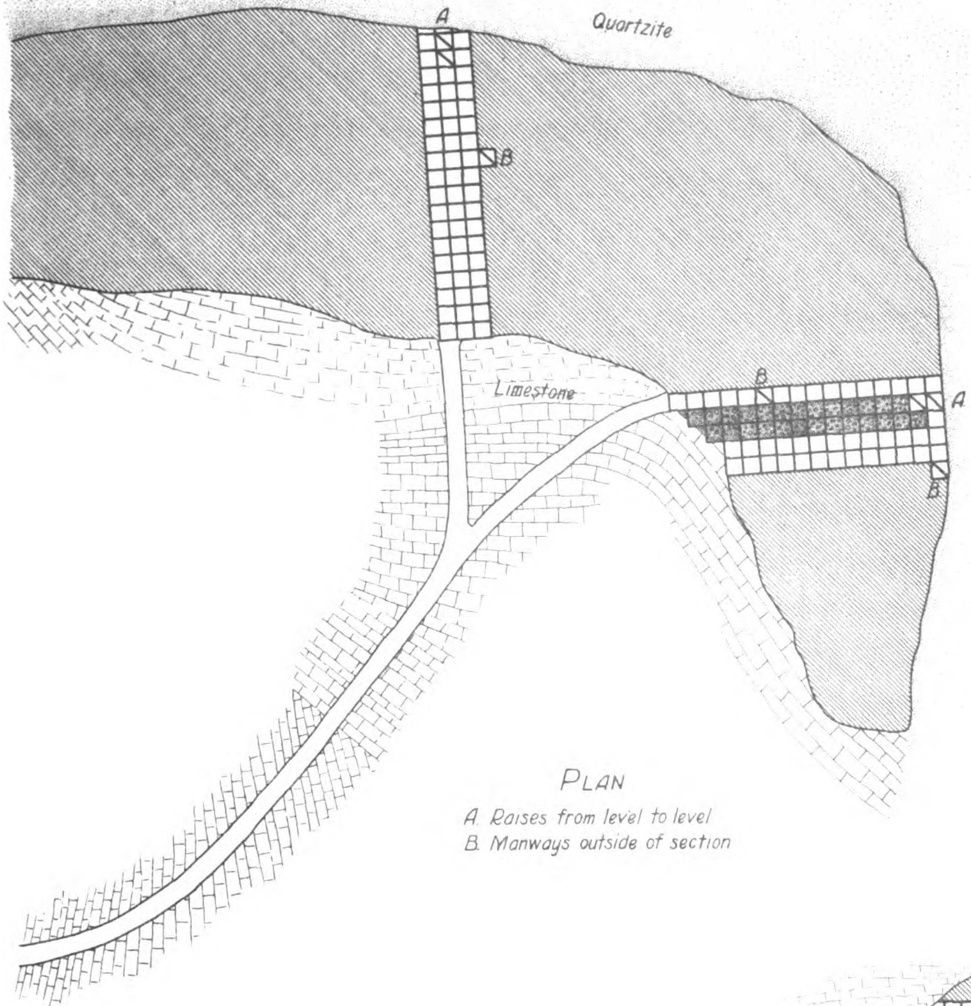
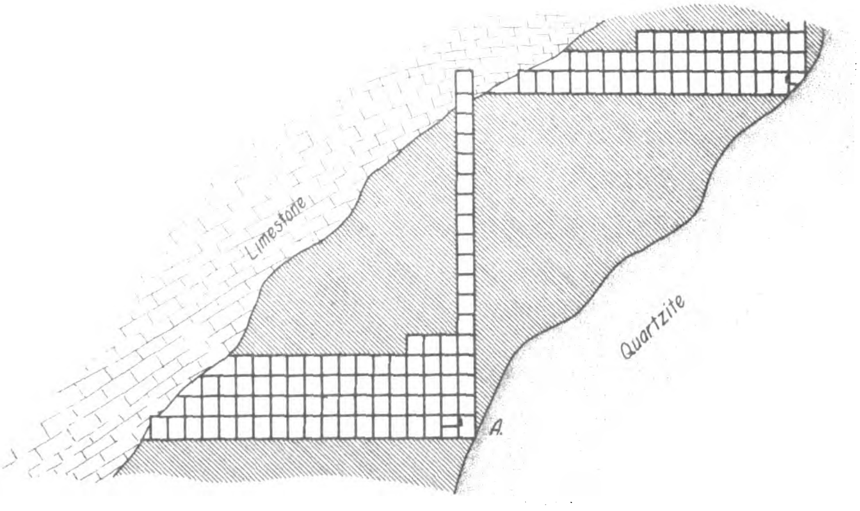

SECTION

Figure 15.— Open cut for waste



PLAN

A. Raises from level to level
B. Manways outside of section



SECTION

Figure 16.– First step in starting a stope

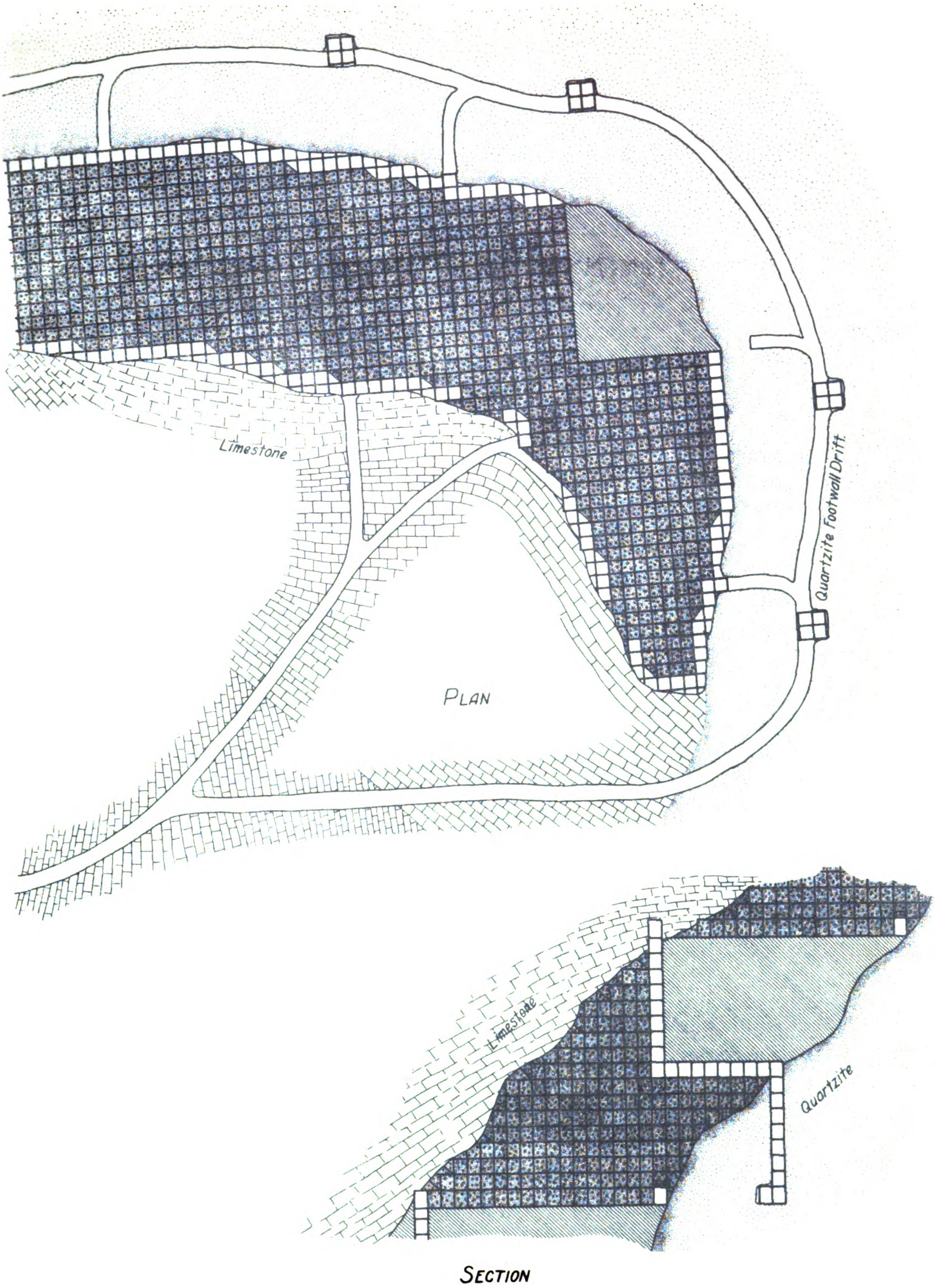


Figure 17.– Method used in mining upper part of a stope

After a number of sections have been mined a filled stope with a vertical face of ore on the footwall end exists. This ore is mined in the same manner, starting a 2-set section from a raise advancing up the footwall. A footwall drift, with raises to the eighth floor (see fig. 17) affords an outlet for the ore above this floor, eliminating the necessity of keeping passages open through filled stopes. The floor of the sections on the footwall are not lagged before filling, there being no ore underneath to be mined from below.

Practically all stopes produce three different types of ore beside waste, and the mine ships these different products separately. Each type of ore is broken separately when possible; otherwise sorting must be done. It is necessary to have more stopes operating than would be needed if only one type of ore were produced, to insure a fixed tonnage of any one product.

The ideal condition for square-set mining is in massive ore that will settle on the timbers gradually. This gradual increase in weight on the timbers as mining is being done tightens the timbers without excessive crushing. Square-set stoping and filling can be done systematically with these conditions prevailing.

Ore bodies of the Tintic Standard are not of the ideal type. In fact, there is nothing regular about the ground, and every kind of condition from solid to material as fine as sand can be expected. The result is that few sections can be mined floor by floor and filled. If ground will not stand while one floor is mined a variation of square-set mining is used, such as working vertical cuts instead of horizontal, working from top down by floors, or working from top down by vertical cuts or combinations of these methods. It is sometimes necessary to make a section only one set wide when the ore is sandy in texture.

A vertical section through the ore body reached a height of 175 feet in some places. Settling of stopes during and for a period after filling must be met. Filling in the stopes will settle as much as 7 feet between levels of 120 feet before they come to rest. This causes great distortion of the timber and settling of the filled stope from the level above, leaving room for movement of the hanging wall. Timbers in the sections mined along the sides of these filled stopes are affected considerably. The shrinkage of the gob in fills can be reduced if wet material is available or if the fill is sprinkled as placed to reduce the voids. These remedies can not be used at this mine for reasons explained later. Cutting caps within the fill from the outside line of posts was experimented with but was not found to be of sufficient value in preventing distortion to be continued as standard practice. It has the further disadvantage of cutting a tie that materially supports the outside line of timbers.

It is not advisable to use water in the fills to the extent that would be necessary to settle the gob. Water reaching the ore removes cementing material in the ore and would make it more troublesome to mine. An additional condition that must be recognized is that early mining was done on the lower levels. It would be dangerous to permit water to percolate through old fills and caves, the movement of which would give much trouble.

The best method found to combat settling under the conditions at this mine is to keep filling as close to mining as possible, allowing the stopes to settle during the period of mining. By so doing it is possible to get the minimum of movement of the fill on the level above, after mining under its sills. A disadvantage of this method is that it allows the ore in the back of the section being mined to crumble, making the ground harder to hold.

Eight by eight inch timbers framed as shown in Figure 18(a) were used during the earlier life of the mine but it was found that they were not as practical for high stopes as timbers of the size and framing shown in Figure 18(b).

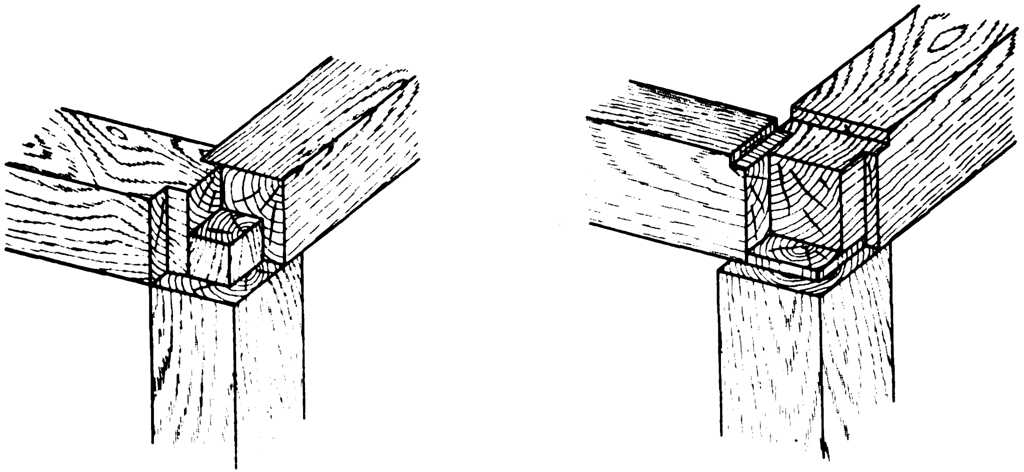
At present the square-set timbers consist of 9 by 10 inch posts, 10 by 10 inch caps, 6 by 10 inch braces, and 6 by 10 inch sills (fig. 18(b)). As the length of the section is from foot wall to hanging wall, caps are placed parallel to the foot wall. This is done for the purpose of having the ends of the caps against the gob so that settling of the fill will not disconnect the timbers at the tenons. The caps have a 5-inch bearing on the posts, while the braces in this framing have only 1 inch. When angle braces are necessary they are always placed from the gob line to the side of the stope, which puts them on the caps (see fig. 19). If the center line of timbers starts riding, diamond braces are used (see fig. 20). This, again, makes use of the caps. Stulls when used are always placed on the caps. In general, when caps are placed at right angles to the gob they are in a position to give the best results when additional timbering must be done, and the strongest member of the set ends against the gob. The gob is always a source of pressure. Figure 19 illustrates different methods of timbering and bracing in a stope against a gob.

Blocking the sets is always done at the joints of timbers directly in line with the end of the caps or braces. No other blocking is allowed until the section is ready for lacing. Just before lacing is nailed on blocks are placed from the center of the outside line of posts to the solid. These center blocks help to keep the posts in place after filling has been placed, especially when one of the joint blocks has dropped out. They are a reinforcement that prevents breaking of the posts from side pressure.

Before taking out the ground to make room for a set along the side of a filled stope, a stringer is placed against the gob posts and braced, as shown in Figure 18, to prevent the gob from moving when the supporting ground is broken. The stringer is 8 by 8 inches by 12 feet long.

There is no horn on the bottom of the post, as miners are disposed to cut it off instead of cleaning out the hole to fit the horn. The cap sets on half of the end of the post, making a good tie from one line of timbers to the next, especially after weight is present to tighten the timbers at connections.

Round poles are used for bridge and stull work, 2 by 12 inch lagging for floors and chutes, and 2 by 6 inch lagging for lacing the corners of the stopes for filling. The flooring is used to lace the remainder of the stope.

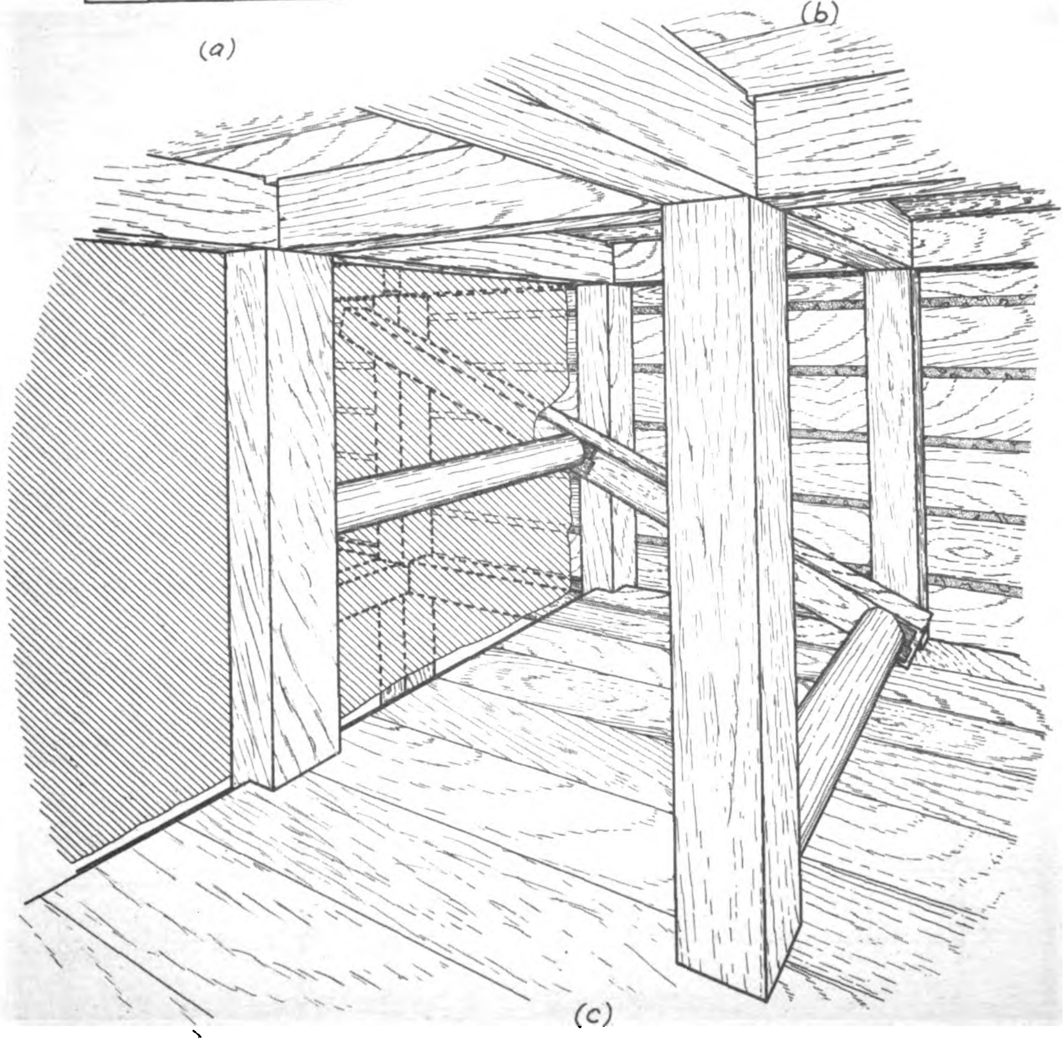


Old-Type Sets
Scale, Inches
0 6 12 24

(a)

Present-Type Sets
Scale, Inches
0 6 12 24

(b)



(c)

Use of Gob Stringer

Figure 18.—Timbering details

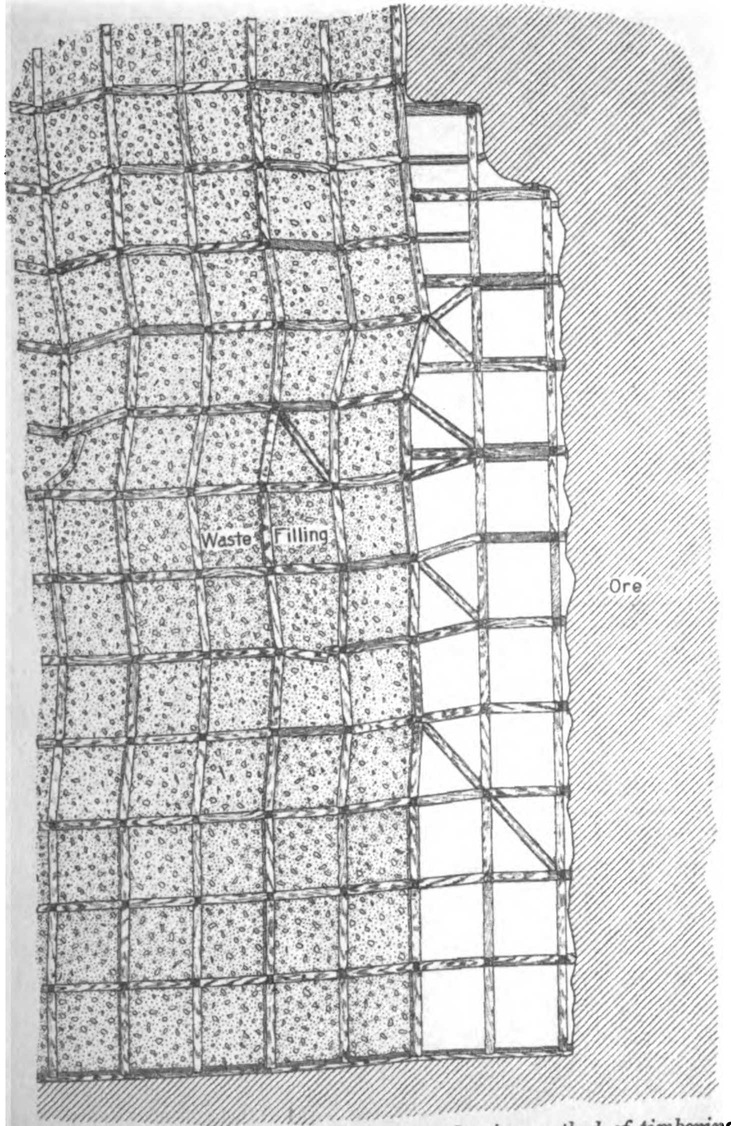


Figure 19. – Vertical section through stope, showing method of timbering

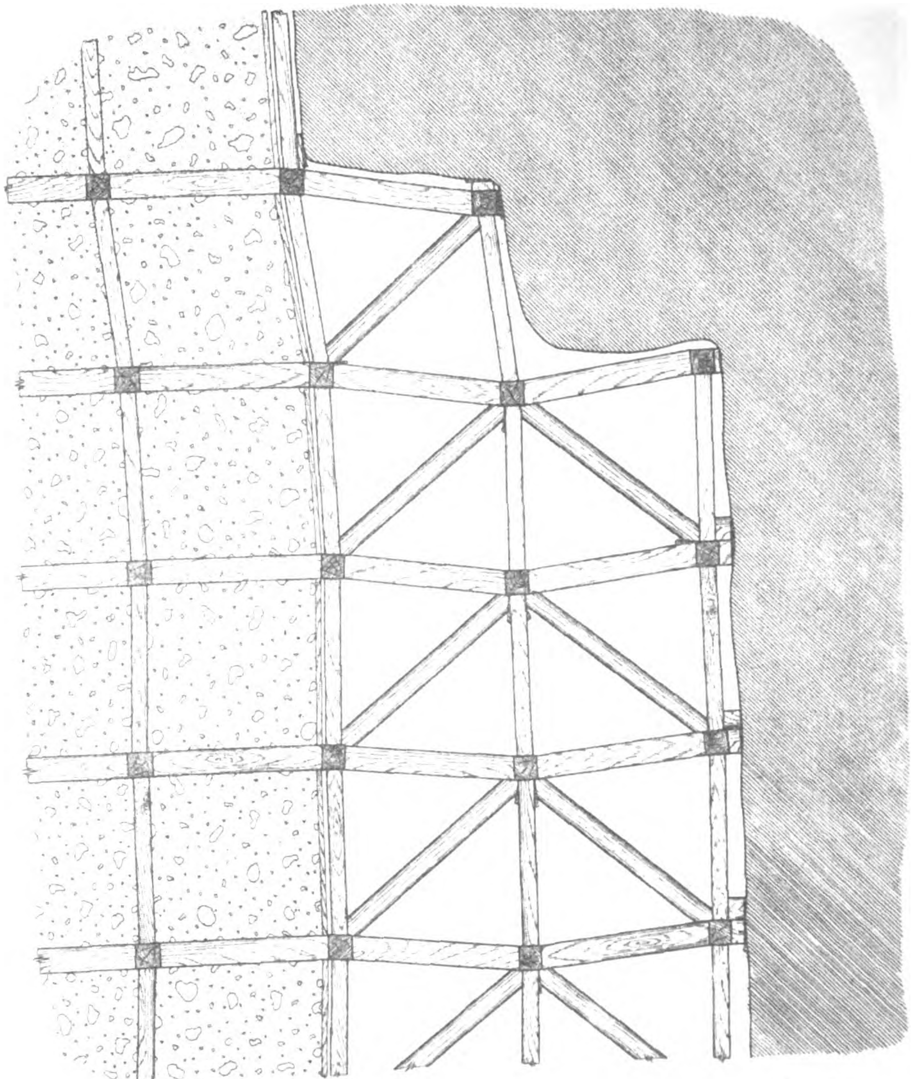


Figure 20. – Horizontal section through stope, showing use of diamond bracing

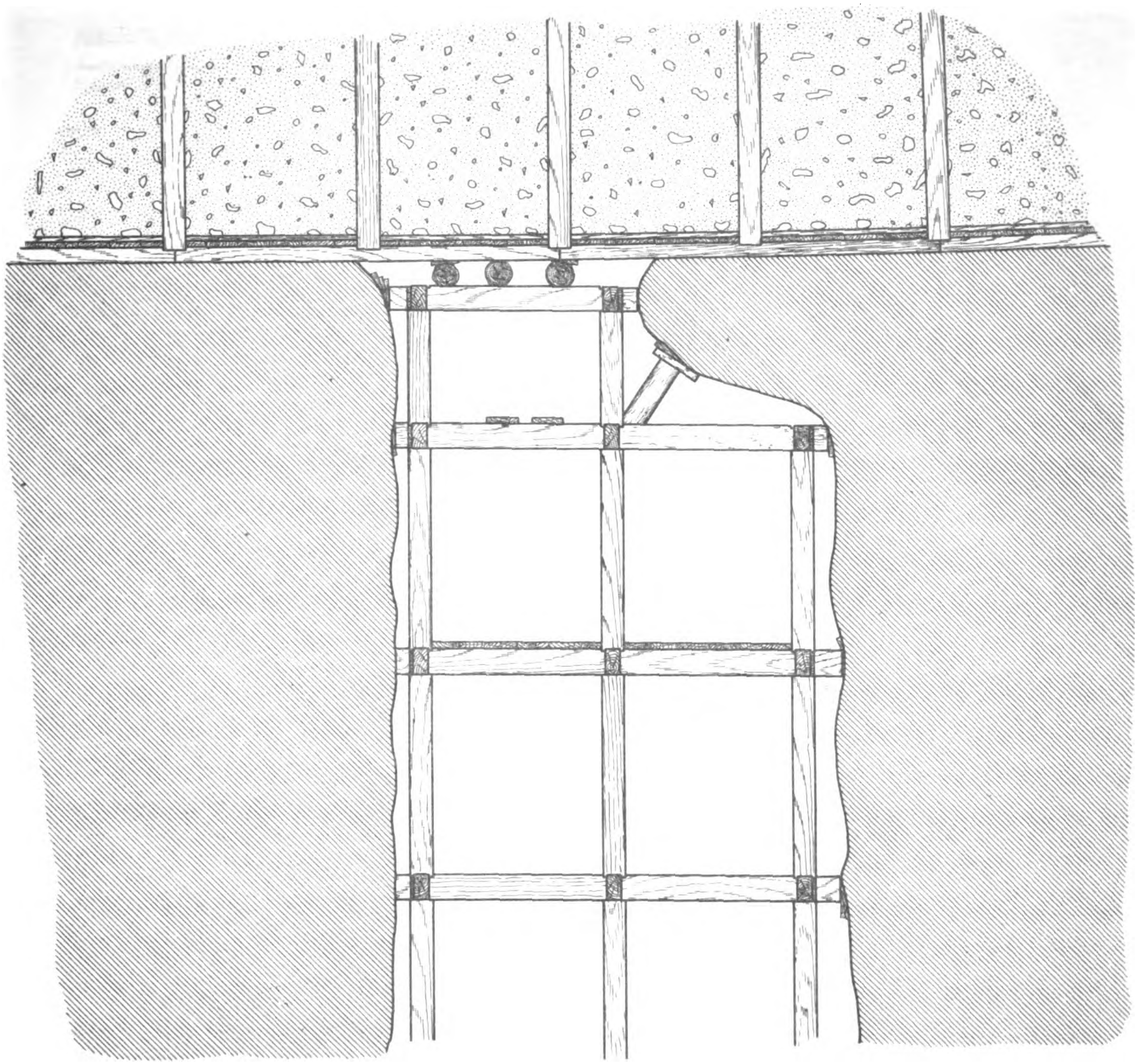
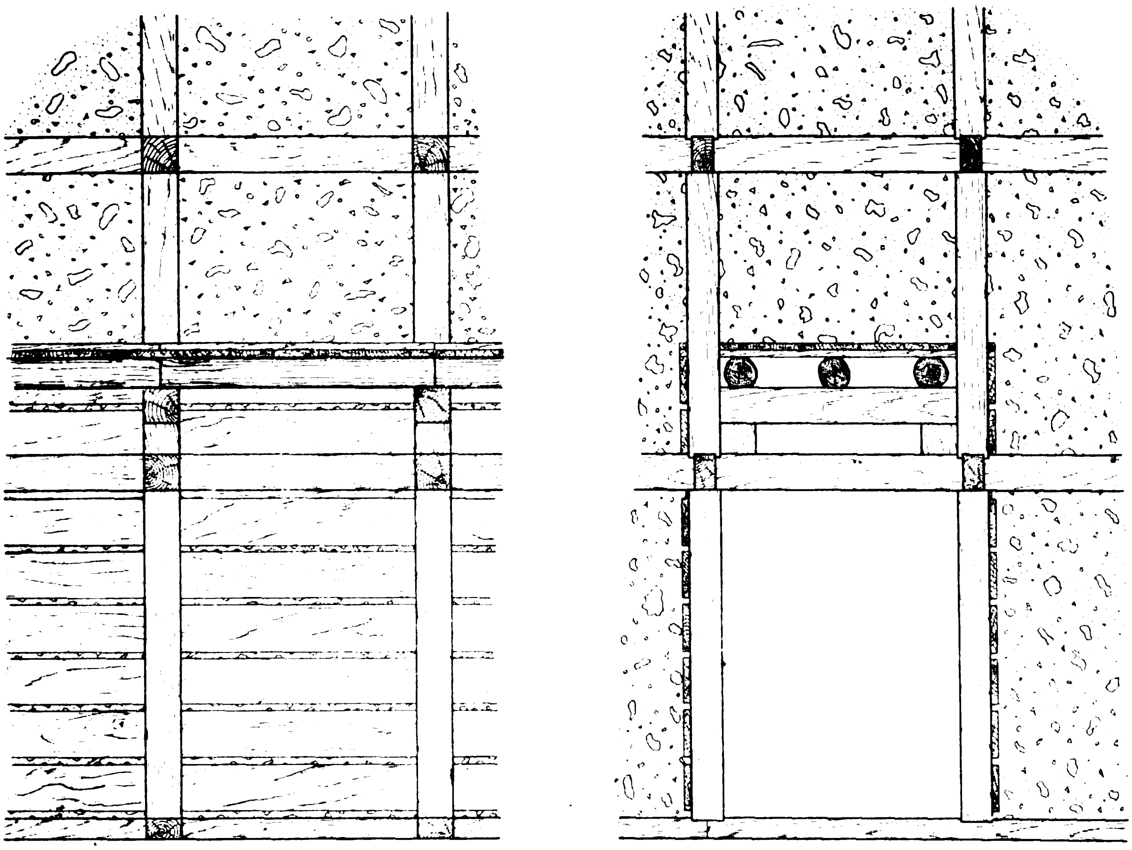


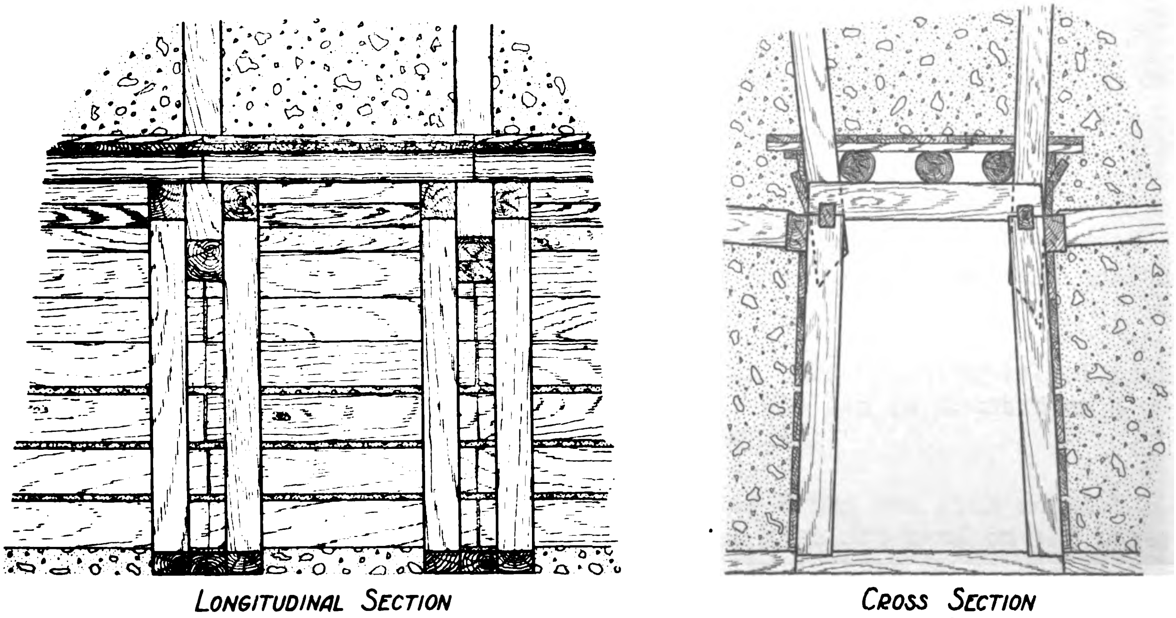
Figure 21. – Section coming up under gob



Longitudinal Section
(a)

Beginning

Cross Section
(b)



Longitudinal Section
(c)

Later

Cross Section
(d)

Figure 22. – Maintaining drift through gob

Corner lacing is shown in Figure 12 (a).

The following method is the best for catching up sills of the stopes when mining is being done underneath. Cut the posts for the sets immediately under the sills so that there is enouch space between the caps and the sills for round poles or square timber to wedge in tightly (see fig. 21). These round timbers lie horizontal from cap to cap and in a position to support the sill underneath the posts above. Naturally, this can be called only a temporary support, as filling must soon follow. By using a method of this sort it is not necessary to attempt to place the square-set stope on the lower level to line up with one on the level above.

Drifts through filled stopes are sometimes a necessity. To keep these drifts large enough without removing any filling when repair work is done bridging is placed on the square sets which allows additional head room (figs. 22 (a) and 22 (b)). After some time the square-set posts in these drifts will break allowing filling to run unless repair work is done. It has been customary to take out the broken post and replace with a new one or place a doubling-up post under the cap. From experience in this mine it has seemed better to place a tunnel set on each side of the set with the broken post, making sure that the tunnel sets support the bridging on top and the gob lacing on the sides (see fig. 22 (c) and 22 (d)). The broken post is then cut out and not replaced. It is better to put the tunnel sets in before the stope posts break, through a gob that must be maintained in good shape for a long period. A direct line of timber extends from the hanging wall through the gob to the stope posts on the sill. When the sill post is cut out the line of posts above will be pushed through the waste fill until they become a part of the fill. After this the only movement of these upper timbers is with the fill itself. It is sometimes necessary to cut the posts on the first floor above the sill two or three times as they push down into the drift.

Care is exercised in blasting in stopes that timber is not broken or shot out. Holes are drilled with stopers or jackhammers.

UNDERGROUND TRANSPORTATION

The main shaft passes through the ore body; consequently distances are short; The ore is trammed by hand. In some of the long development drifts mules are used.

The cars are 18 cubic feet in capacity and hold approximately 1 ton. One end of the car box is a door, swinging outward; the box is mounted on a turntable so that the cars are both end and side dumping.

Cages are double decked; either one car or 8 men can ride on each deck. When men are riding solid-iron doors 4½ feet high are used on the cage; to hold the cars on the cage there are short right-angle bars on either side which turn down both in front and behind the car.

Cage riders are the only ones allowed to use the electric signals to the hoist engineer; others call the cager by telephone (there is one on each level). The cager and helper load all cars underground onto the cage and remove the empties from the cage coming down. On the surface the top-carmen push off the loaded cars, replacing them with empties, and hand tram the ore or rock to the top of the ore bins or the dump. The ore from the surface bins is loaded directly into standard-gage railroad cars.

PER CENT EXTRACTION

The loss of ore and the dilution of ore by waste in this mine are negligible. So far a very small amount has been lost by caving, and a trifle may be left here and there on the hanging or footwall, but the combined loss is probably less than 1 per cent. No sorting is done on the surface and underground only an occasional bowlder of waste is thrown out.

ORGANIZATION

Figure 23 shows the complete organization and the average number of men supervised by each. This organization, with the exception of the underground force, also serves four other properties, namely, Eureka Standard, Iron King, Colorado Consolidated, and Iron Blossom.

WAGE, CONTRACT, AND BONUS SYSTEMS

Contracts here are virtually bonus systems. So-called contracts consist of verbal agreements to guarantee wages and also to pay a stipulated price per foot for direct labor. These contracts are let only on drifts, raises, shafts, and winzes and are really negligible; contracts let in 1929 amounted to less than 1 per cent of the pay roll and in 1928 there were none.

The following wage scale went into effect on March 10, 1930. Prior to that time underground wages were 75 cents per day greater and surface wages 50 cents per day greater.

Shift bosses	$6.25
Special timbermen	5.25
Special timbermen helpers	5.00
Timbermen	4.75
Timbermen helpers	4.50
Shaft miners	5.50
Shaft miner helpers	5.25
Miners	4.75
Muckers	4.25
Trammers	4.25
Cagers	4.75
Cager helpers	4.50
Pipe, track, and ventilation men	4.75
Pipe, track, and ventilation man helpers	4.50

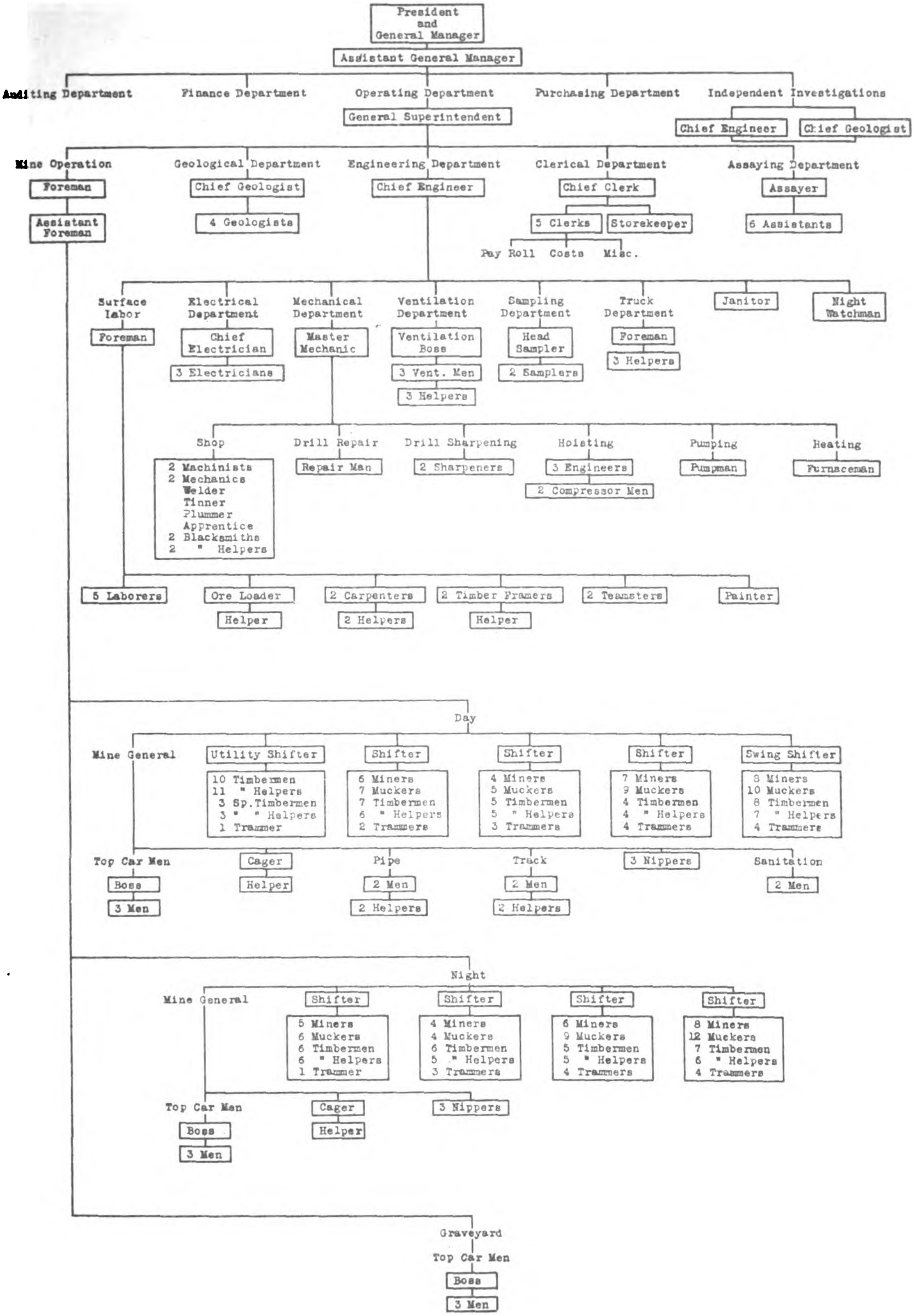
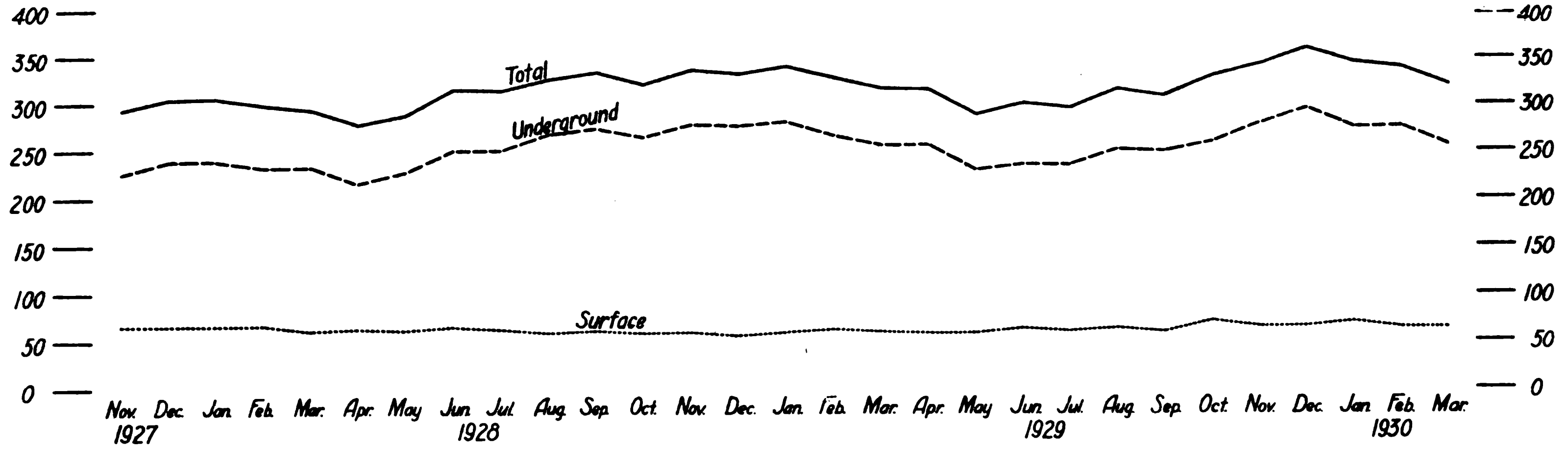
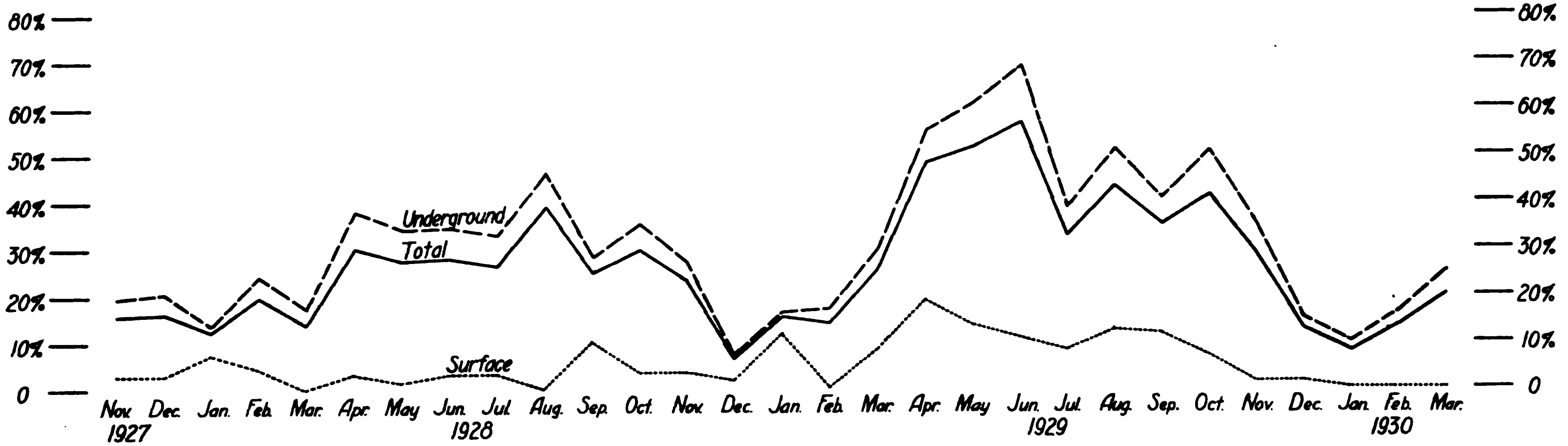


Figure 23. – Organization chart



MEN EMPLOYED



LABOR TURNOVER

Figure 24. — Number of men employed and monthly labor turnover

Tool nippers	$4.75
Top-carmen	4.75
Top-carman helpers	4.25
Hoist engineers	5.75
"Donkey" hoist engineers	4.75
Compressor men	4.50
Machinists	5.25
Welders	5.25
Pipe fitters	4.50
Blacksmiths	5.25
Blacksmith helpers	4.00
Electricians	4.50 to 5.50
Drill repair men	5.50
Drill sharpeners	5.25
Carpenters	5.50
Carpenter helpers	4.25
Timber framers	4.25
Teamsters	4.75
Laborers	3.75
Fuse men	4.50

Figure 24 shows the labor turnover in per cent, computed from the monthly men hired divided by the average number of men maintained.

VENTILATION

The Tintic Standard mine is located in a high-temperature area. In virgin rock the average temperature is about 103° F., the maximum temperature recorded being 126° F. Ventilation is mechanical throughout. The principal object of ventilation is to lower the temperature and cause a positive circulation of air through the mine. The average mine temperatures are 79° F. dry bulb and 74° F. wet bulb, relative humidity being 79 per cent.

The primary circuit of air is controlled by six large fans handling about 100,000 cubic feet per minute through the mine by way of five shafts (including the North Lily and Eureka Standard) and supplying air for the secondary circuits. The secondary circuits consist of 32 small fans which pick up air from the primary circuit and deliver it to the various working places. Blowers have been found to be more efficient than exhausters, the principal reasons being: First, recirculation is prevented by better control of the air currents; second, gas seems to be retained in the rock better when air is blown into the workings than when the air is exhausted. Carbon dioxide and nitrogen are encountered at times. During early development of the mine much time was lost from gas. Weather conditions or the barometric pressure controlled the working time when only natural ventilation was used. Exhausting fans were first installed, and conditions were greatly improved. As the mine developed these were changed to blowers, which resulted in even greater improvement in the comfort of the working places.

The primary circuit, as shown in Figure 9, is roughly divided into three parts. Although these three circuits are not entirely independent of each other, they are classed as such.

The No. 2 shaft, which is the main working shaft of the Tintic Standard, is located very nearly in the center of the main ore body. This shaft is downcast and is connected with all the main working levels. This shaft is an ideal supply for the fans of the secondary system; electrically operated doors are located on the various levels between the fans and the working places to prevent recirculation. About 40,000 cubic feet per minute is taken down the No. 2 shaft, the larger portion of which goes to the lower levels of the mine and is discharged through the No. 1 shaft by a 40-hp. single-inlet fan handling 24,500 cubic feet per minute on the 1,000 level and a 50-hp. single-inlet fan handling 24,000 cubic feet per minute on the 1,300 level. A small portion of the air leaves the mine by way of the No. 3 shaft and Eureka Standard shaft (see section A-A', fig. 9).

The North Lily shaft and mine, about 3/4 of a mile northwest of the Tintic Standard, is downcast and supplies 40,500 cubic feet per minute through the two connections to the Tintic Standard mine. The North Lily mine is located in a comparatively low-temperature area, so that the air after passing through it is still desirable both as to temperature and himidity. A 30-hp. single-inlet fan delivers 21,000 cubic feet per minute on the 900 level, and a 20-horsepower double inlet fan delivers 19,500 cubic feet per minute on the 700 level. This air ventilates the northeast section of the mine and leaves the mine by way of the No. 3 shaft, where a 60-hp. single-inlet fan is located at the collar and discharges 31,000 cubic feet per minute. Some of the air from this circuit leaves the mine by way of the Eureka Standard shaft, also a small amount of air works through into the No. 1 circuit and is discharged via the No. 1 shaft (see section B-A' of fig. 9).

The third circuit shown by section B-B' of Figure 9 is controlled by a 30-hp. double-inlet fan on the 900 level which picks up excess air from the other two circuits and discharges about 20,500 cubic feet per minute through a 5,000-foot connection to the Eureka Standard shaft at the 800 level. This shaft acts as an upcast for the Tintic Standard and as both upcast and downcast for the Eureka Standard mine. This shaft is 6 feet wide by 22 feet long, the hoisting compartments on the south end being separated from the ventilation compartment and manway by a solid partition. At the collar a 100-hp. single-inlet fan handling 76,500 cubic feet per minute is connected to the upcast compartment. Fresh air is drawn down into the mine through the hoisting compartments, which have been gunited to prevent short-circuiting. The fresh air is picked up on each working level by secondary fans, which force it to the various working faces. The return air works up through the various raises to the 1,000 level, which is connected to the upcast compartment.

Secondary ventilation is furnished in the Tintic Standard by small, high-pressure fans. Twenty-four of the thirty-two secondary fans are electric, using 5 to 15-hp. each, and a total of 225-hp. The remainder of the secondary fans are pneumatic. Sixteen inch, 20 gage galvanized metal pipes are used

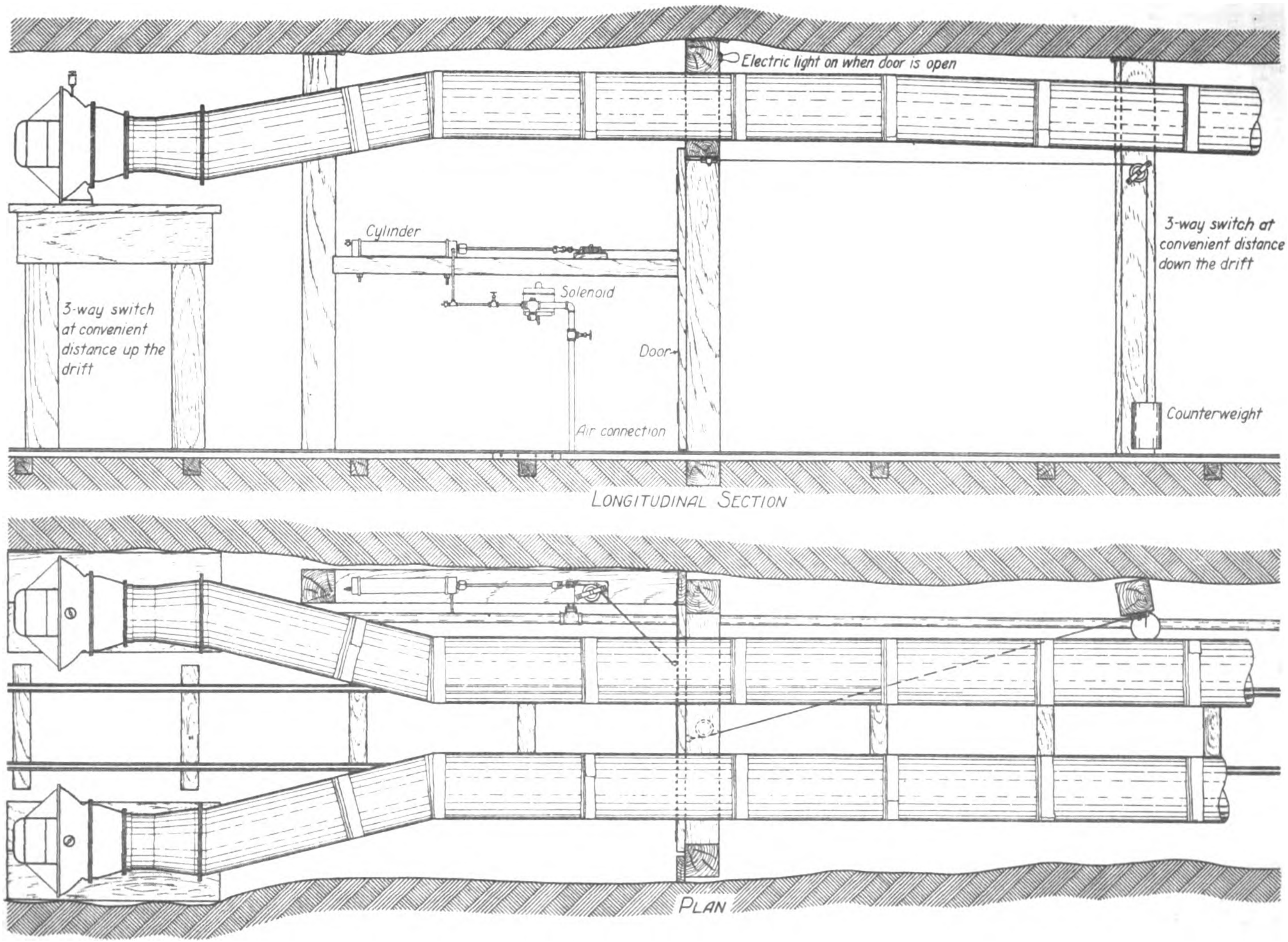

Electric light on when door is open
Cylinder
Solenoid
3-way switch at convenient distance up the drift
Door
Air connection
3-way switch at convenient distance down the drift
Counterweight
LONGITUDINAL SECTION
PLAN

Figure 25. – Air door and secondary fan installation

almost entirely for ventilation lines, the joints of which are well wrapped and painted to prevent leakage. Some flexible tubing is used at the ends of the metal lines to insure rapid removal of ventilation line before blasting.

Electrically operated doors have been used satisfactorily for a number of years, and control air currents very efficiently. Figure 25 shows a typical door and secondary fan installation. In many cases doors can be placed near the stopes, thus shortening ventilation lines materially. Whenever possible secondary fans are installed in fresh air or on the primary circuit side of the door.

A typical primary fan installation is shown in Figure 26. Double doors are a part of each installation; these allow traffic past a fan without interrupting the circuit.

Ventilation has always been one of the major problems at the Tintic Standard mine. At present, working conditions are much better than ever before, due to the increased number of connections between levels, the number of shaft connections, the number of shafts connected, and the amount of air forced through the mine. To maintain satisfactory working conditions, however, the ventilation system must be watched continually, air currents controlled at all times, and ventilation lines maintained as close to the working faces as possible. Should an interruption in power shut down the fans for but a short time the men are forced to leave their working places.

The following chart shows the distribution of air through Tintic Standard mine:

Ventilation chart

Into mine	Out of mine
No. 2 shaft...............41,000 c.f.m.	No. 1 shaft...............48,500 c.f.m.
North Lily shaft..........40,500 c.f.m.	No. 3 shaft...............31,000 c.f.m.
Expansion...6,500 c.f.m.	Eureka Standard shaft.....20,500 c.f.m.
Compressed air.......1,000 c.f.m.	
Downcast fissure in mine....11,000 c.f.m...18,500 c.f.m.	
Total..................100,000 c.f.m.	Total..................100,000 c.f.m.

Horsepower rating of ventilation fans

Primary circuits......................	330 Hp.
Secondary circuits....................	225 "
Total.................................	555 "

All motors are fully protected by magnetic switches and are equipped with stop and start push buttons, giving remote control when desired.

FIRE HAZARDS

The fire hazard at the Tintic Standard is quite appreciable. The main primary air inlet, which is the main working shaft, is well timbered. The stations are in heavy ground and have to be heavily timbered. In fact, there are very few places in the whole mine where an abundance of timber is not necessary. The mine is hot and dry, necessitating strong air currents.

To prevent fires, these conditions require a great deal of care and caution, which are exercised. Only one very small fire has ever occurred. This was of unknown origin and did no damage except for the excitement of a little smoke.

Surface buildingsare iron clad and as far from the shaft as is practical. No smoking or open lights are allowed on the cages or near the shaft, both on surface or underground. Electric wiring consists of heavily insulated cable and is under constant observance against damage from sagging timber. Ventilating fans and motors are generally placed back in the drifts, well away from the stations; where this is not practical the fan rests on an iron plate covered with sand to absorb the oil spilled, and the adjacent timbers are well gunited. Two men are employed for keeping the stations and drifts clear of rubbish and also for sanitary purposes. There are signs, in English and in Spanish, on all powder magazines forbidding open lights. These magazines are also kept under lock and key by the shift boss until just before blasting time.

Special attached fire hose are kept on each station, and water that can be used on fires is piped to all working faces.

The ventilation system is planned with a view to the control of air currents in case of fire, so that emergency outlets can be kept free of smoke while the men are leaving. The system is also planned to control the fire by restriction and to aid in the performances and protection of fire fighters.

A complete outfit of five self-contained breathing apparatus of the Gibbs type, with the necessary auxiliary equipment, is kept on the surface within 100 yards of the shaft. These apparatus and accessories are kept in perfect repair, being inspected semimonthly by a thoroughly competent man. There are about 35 men employed, who have been trained in its use during the periodic visits of the U. S. Bureau of Mines mine rescue car.

The surface yards are protected by equipped high-pressure hydrants and by a night watchman.

SAFETY

The management believes in the economy of safety. It follows, then, that in planning future developments accident prevention receives major consideration, and that the safety idea permeates the whole organization.

Safety posters and bulletins daily remind the men to be on guard. New men are given a book of instructions and general safety rules when hired. Blasting is permitted only at regular specified times, twice a shift. Primers are made up on the surface and sent to each level in individual heavy fiberboard containers. Cages and hoisting equipment are rigidly inspected and oiled daily; in fact, guards for all machinery get the same treatment. Failure to replace guards at the shaft and in the raises means immediate dismissal.

All emergency outlets are plainly marked and it can be seen by the foregoing maps that they are in abundance.

The foreman and shift bosses hold regular monthly meetings to discuss accidents that have occurred, ways and means to prevent their reoccurrence, and current safety practices.

In June, 1928, a safety bonus system based on the minimum lost time per man-shift was started for shift bosses. It consists of three yearly cash awards, namely: Fifty, thirty and twenty dollars each. Monthly ratings are published in the shift bosses' change house. The bonus has had the effect of eliminating the careless or incompetent workman and has reduced the number of accidents. For instance, in 1927, the year before, there were 8,286 days lost by workmen due to accidents, while in 1929 there were only 1,571 days lost. These days lost are according to the Government's ratings of accidents. In 1927 there was one fatal accident, which accounts for 6,000 days, but even excluding this the accidents were only 73 per cent of those of 1927.

First-aid training is given by the U. S. Bureau of Mines car to those who care to take it. This knowledge is appreciated by the company, but it is to be practiced only when the doctor is not immediately available. A company doctor and an emergency hospital are maintained, and the doctor goes underground and personally takes care of any accident. Only in exceptional emergency is first aid practiced by the miner.

Frequent visits and suggestions of the State mine inspector are appreciated and encouraged.

SUMMARY OF COSTS

Operating costs are shown in Tables 1 and 2, the actual cost statement covering the month of November, 1929. Included in the total cost is the expenditure that is made to maintain and serve the company's town of Dividend. Those items, which are direct charges, have been segregated into stoping, filling, and development.

Table 1.--Summary of costs

Name or number of mine - Tintic Standard. Period covered - November, 1929.
Tons ore hoisted during period - 8,758 dry tons
Mining method - Square set and fill.

Underground costs per dry ton of ore hoisted	1	[a]2	[b]3	4	5	6	7	8
Development								
In ore	$0.184	$0.020	$0.062	---	$0.022	$0.007	$0.004	$0.299
In rock	.508	.054	.134	---	.087	.051	.010	.844
Mining								
Stoping	2.109	.223	.521	---	.099	.691	.049	3.692
Maintenance	.650	.069	.053	---	.015	.150	.012	.949
Filling	.291	.031	---	---	---	---	.010	.332
Transportation (underground)	.752	.079	---	$0.135	---	---	.090	1.056
Underground general								
Ventilation	.106	.011	---	.080	---	.008	.039	.244
Track	.070	.007	---	---	---	---	.025	.102
Sampling	.036	.004	---	---	---	---	.001	.041
General	.034	.003	---	---	---	---	---	.037
Surface								
Mine general	.425	.045	---	---	---	---	.293	.763
Unloading	.045	.004	---	---	---	---	---	.049
Yard maintenance	.086	.009	---	---	.001	.003	.038	.137
Load and shipping	.029	.003	---	---	---	---	---	.032
Assaying	.063	.007	---	---	---	---	.041	.111
Mine office	.119	.013	---	---	---	---	.010	.142
Team, auto and truck	.031	.003	---	---	---	---	.091	.125
Building maintenance	.097	.010	---	---	---	.003	.138	.248
Heat, water, and light	.096	.010	---	.046	---	---	.109	.261
Powder man	.018	.002	---	---	---	---	---	.020
Totals	5.749	.607	.770	.261	.224	.913	.960	9.484

Columns 1 to 8 represent:

1. Direct labor
2. Supervision ([a]distributed on direct labor basis)
3. Compressed air, drill, and steel ([b]distributed on machine shifts)
4. Power cost
5. Explosives
6. Timber
7. Other supplies
8. Total

Table 2.--Summary of costs in units of labor, power, and supplies

Name of mine - Tintic Standard. Period covered - November, 1929.
Tons ore hoisted and mined during period - 8,758 dry tons.
Mining method - Square set and fill.

	Development		Mining			Total
	In ore	In rock	Stoping	Maintenance	Filling	
A. Labor(man-hours per ton of ore)						
Breaking	$0.0969	$0.2133	$0.7333	$0.0349	---	$1.0784
Timbering	.0955	.1805	.6852	1.4095	$0.1910	2.5617
Mucking	.0948	.2558	.8268	.0725	.2942	1.5441
Tramming	.0369	.0934	.3123	.0119	.2412	.6957
Supervision [1]	.0270	.0744	.3089	.0952	.0426	.5481
General [1]	.0393	.1082	.4491	.1384	.0619	.7969
Total labor underground	.3904	.9256	3.3156	1.7624	.8309	7.2249
Average tons per man per shift						1.1073
Underground labor, per cent of total cost	3.1	8.5	35.2	10.8	4.8	62.4
Average tons per man-shift on surface properly chargeable to underground operations						7.008
B. Power and supplies						
Explosives(pounds per ton)[2]	.1058	.4618	.4184	.0816	---	1.0676
No. 8 caps per ton	.0862	.2382	.6743	.0403	---	1.0390
Fuse, feet per ton	.5073	1.4336	3.5342	.2224	---	5.6975
Timber (b. m. per ton)	.0200	1.4571	19.7428	4.2857	---	25.5056
Total power (kw.h. per ton)	2.84	4.41	32.45	5.63	1.27	46.60
Air compression [1]	.80	2.19	9.11	2.80	---	14.90
Hoisting [1]	1.24	---	14.16	---	---	15.40
Ventilation [1]	.57	1.57	6.54	2.02	.90	11.60
General, lighting, etc. [1]	.23	.64	2.65	.82	.36	4.70
Other supplies, per cent of total supplies and power						29.7
Supplies and power, per cent of total cost						31.6
C. Per cent of total costs	4.9	13.8	60.4	15.5	5.4	100.0

[1] Distributed on direct labor basis.

[2] Represents both 25 per cent and 40 per cent gelatin dynamite in very nearly equal portions.

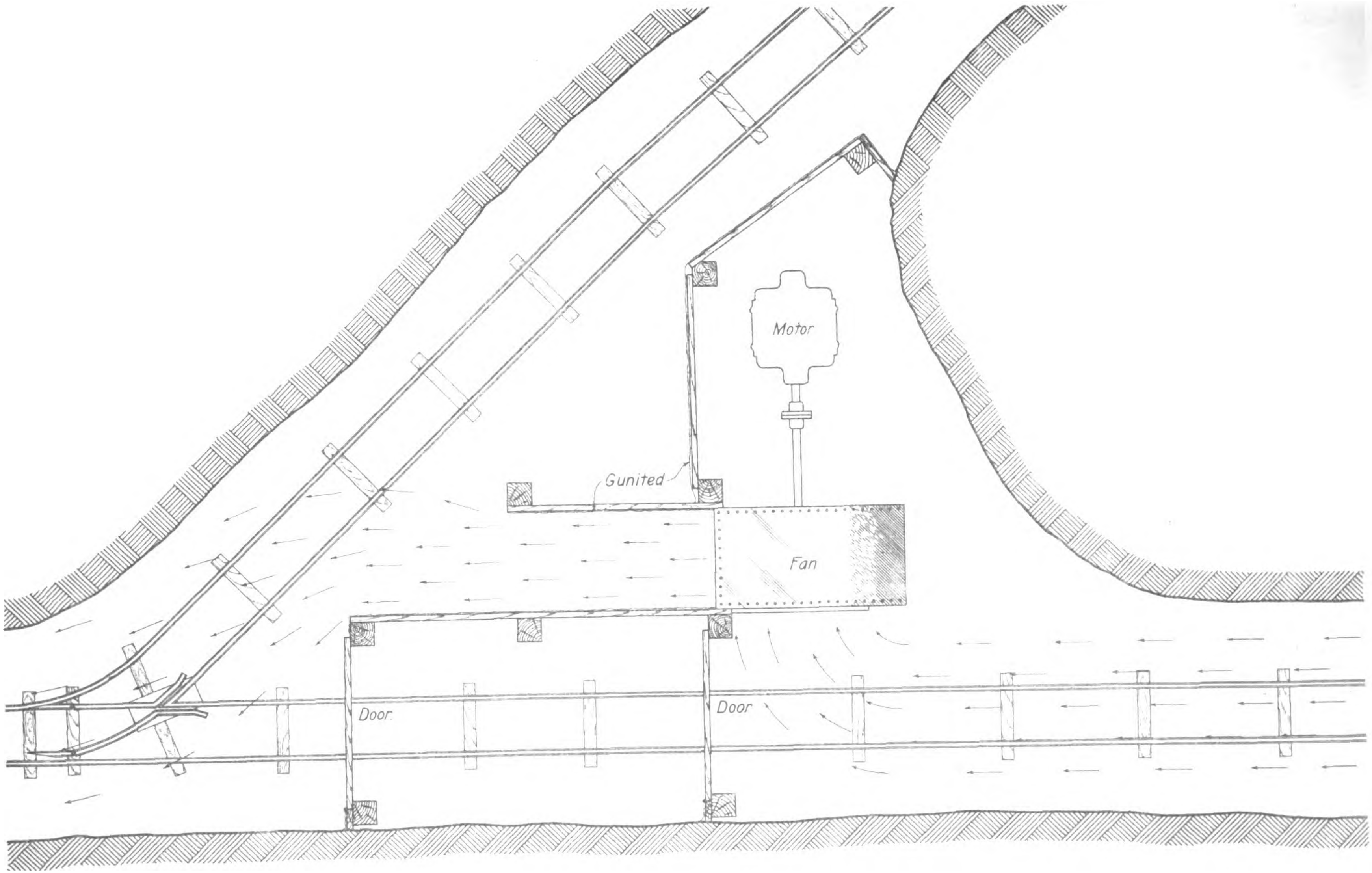


Figure 26. – Typical underground primary exhaust fan installation